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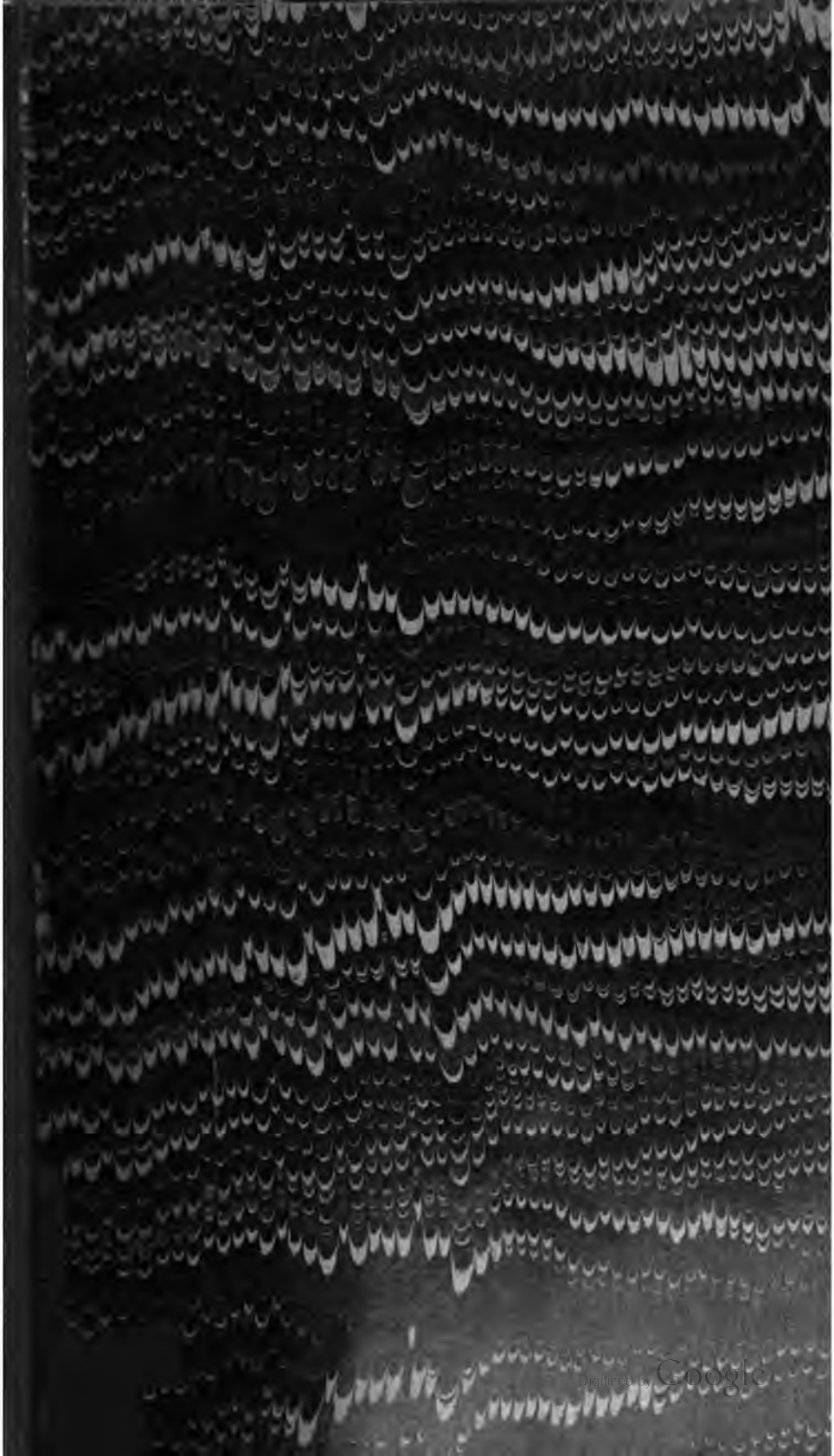
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J. C. Branner.

TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS.

VOL. XLIV.

CONTAINING THE PAPERS AND DISCUSSIONS OF THE CLEVELAND MEETING, OCTOBER, 1912.

NEW YORK, N. Y.:
PUBLISHED BY THE INSTITUTE,
AT THE OFFICE OF THE SECRETARY.
1913.

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PREFACE.

THE papers and discussions contributed to the Institute during the year 1912 were so many and so extensive that it was impracticable to include all in one regular volume without increasing the size beyond reasonable limits. It was therefore decided to issue two volumes for the publications of 1912.

Vol. XLIII., containing the papers and discussions of the New York meeting of February, 1912, and some papers omitted from the preceding volume for lack of space, as well as such of the papers presented at meetings of the Local Sections of the Institute as were furnished for publication, was issued to the membership in May, 1913. The reports of the Board of Directors and the Council, lists of officers, statistics of membership, and other official announcements which are published annually for the information of the members, are included in that volume.

The present volume, XLIV., contains the papers and discussions of the Cleveland meeting of October, 1912, and one paper, Temperature Conversion Tables, by Dr. Leonard Waldo, which was presented by title at the New York meeting, February, 1912, but was not finally approved by its author in time to be included in Vol. XLIII.

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CONTENTS.

PROCEEDINGS.

Cleveland Meeting, October, 1912, ... vii

PAPERS.

PAGE.

The Iron-Ores of the South Range of the Cuyuna District, Minnesota. By CARL ZAPFFE and W. A. BARROWS, JR., ... 3
A Titaniferous Iron-Ore Deposit in Boulder County, Colo. By E. P. JENNINGS, 14
Clinton Iron-Ore Deposits in Kentucky and Tennessee. By S. WHINERY (see Discussion, p. 889), ... 25
The Ta-yeh Iron-Ore Deposits, Hu-pei Province, China. By C. M. WELD, . 27
The Concentration of Iron-Ores. By N. V. HANSELL (with Discussion), . . 37
Surveying and Sampling Diamond-Drill Holes. By E. E. WHITE, . . . 69
New Type of Blast-Furnace Construction. By J. E. JOHNSON, JR., . . . 91
Blowing-in a Blast-Furnace. By R. H. SWEETSER (with Discussion), . . 105
The Effect of Alumina in Blast-Furnace Slags. By J. E. JOHNSON, JR. (with Discussion), ... 123
The Utility of Efficiency-Records in the Manufacture of Iron. By JOHN JERMAIN PORTER, ... 143
The Manufacture of Coke. By WILLIAM HUTTON BLAUVELT, ... 154
The Manufacture of Coke. By F. E. LUCAS, ... 170
By-Product Coke. By C. W. ANDREWS, ... 181
The Manufacture of Coke. (A Discussion), , ... 184
Fuel-Efficiency of the Cupola-Furnace. By JOHN JERMAIN PORTER, . . 201
Melting Iron in the Cupola-Furnace. By Dr. R. MOLDENKE, ... 211
The Methods of the United States Steel Corporation for the Commercial Sampling and Analysis of Pig-Iron. By J. M. CAMP, ... 220
Recent Developments in Open-Hearth Steel-Practice. By N. E. MACCALLUM, . 250
Methods of Preparing Basic Open-Hearth Steel for Castings. By H. F. MILLER, JR., ... 263
Recent Developments in the Inspection of Steel Rails. By ROBERT W. HUNT (with Discussion), ... 269
Notes on Titanium and on the Cleansing Effect of Titanium on Cast-Iron. By BRADLEY STOUGHTON (with Discussion), ... 282
The Effect of High Carbon on the Quality of Charcoal-Iron. By J. E. JOHNSON, JR. (with Discussion), ... 314
Measurements and Relations of Hardness and Depth of Carbonization in Case-Hardened Steel. By MARK A. AMMON (with Discussion), ... 356
The Action of Various Commercial Carbonizing-Materials. By ROBERT R. ABBOTT (with Discussion), ... 371
Note on the Case-Hardening of Special Steels. By ALBERT SAUVEUR and G. A. REINHARDT (with Discussion), ... 422
Notes on Ruff's Carbon-Iron Equilibrium Diagram. By HENRY M. HOWE (with Discussion), ... 430
Wittorff's Iron-Carbon Equilibrium Diagram. By BRADLEY STOUGHTON, . 478
The Microstructure of Iron and Steel. By WILLIAM CAMPBELL, ... 489
The Influence of Divorcing Annealing on the Mechanical Properties of Low-Carbon Steel. By HENRY M. HOWE and ARTHUR G. LEVY, ... 513
Electric Heating and the Removal of Phosphorus from Iron. By ALBERT E. GREENE, ... 548
The Function of Slag in Electric Steel-Refining. By RICHARD AMBERG, . . 557

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Heat-Losses in Furnaces. By F. A. J. FITZGERALD, ... 569
Alloys of Cobalt with Chromium and Other Metals. By ELWOOD HAYNES (with Discussion), ... 573
The St. Helens Mining-District. By HORACE V. WINCHELL, ... 580
The Occurrence of Gold in the Eocene Deposits of Texas. By E. T. DUMBLE, 588
Present Conditions of Mining in the District of Vladivostok, Siberia. By ALBERT F. J. BORDEAUX, ... 592
A Trip Through Northern Korea. By HENRY W. TURNER, ... 603
Our National Resources and Our Federal Government. By R. W. RAYMOND (with Discussion), ... 612
Fires in Metalliferous Mines. By GEORGE J. YOUNG, ... 644
The Ultimate Source of Metals. By BLAMEY STEVENS, ... 663
Centrifugal Machines for Ore-Grading and Ore-Concentrating. By GODFREY T. VIVIAN (with Discussion), ... 676
The Wood Flotation Process. By HENRY E. WOOD, ... 684
An Experience in the Use of Water-Power. By C. M. MYRICK, ... 702
A Graphic Solution of D'Arcy's Formula for the Transmission of Compressed Air in Pipes. By NATHANIEL HERZ, ... 705
Notes on Bag-Filtration Plants. By A. EILERS, ... 708
Development of the American Water-Jacket Lead Blast-Furnace. By R. C. CANBY (see Discussion, p. 890), ... 736
Development of the Parkes Process in the United States. By ERNST F. EURICH, 741
The Constitution and Melting-Points of a Series of Copper-Slags. By CHARLES H. FULTON, ... 751
The Development of the Reverberatory Furnace for Smelting Copper-Ores. By E. P. MATHEWSON, ... 781
Chemistry of the Reduction Processes in Use at Anaconda, Mont. By FREDERICK LAIST, ... 806
The Sulphatizing-Roasting of Copper-Ores and Concentrates. By UTLEY WEDGE, 818
Notes on the Metallography of Alloys. By WILLIAM CAMPBELL. . . . 825
The Alundum Extraction-Thimble Used in the Determination of Copper. By L. W. BAHNEY, ... 851
The Sampling of Gold-Bullion. By FREDERIC P. DEWEY (with Discussion), 853
Temperature Conversion Tables. By LEONARD WALDO (with Discussion), . 883

DISCUSSIONS.

Of Mr. Whinery's Paper on Clinton Iron-Ore Deposits in Kentucky and Tennessee (see p. 25), ... 889
Of Mr. Canby's Paper on Development of the American Water-Jacket Lead Blast-Furnace (see p. 736), ... 890

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Proceedings of the One Hundred and Third Meeting, Cleveland, Ohio, October, 1912.

LOCAL COMMITTEE.

EXECUTIVE.—D. T. Croxton, *Chairman;* C. B. Murray, *Secretary;* F. B. Richards, W. G. Mather, A. W. Smith, S. T. Wellman.

FINANCE.—W. G. Mather, *Chairman;* H. H. Brown, C. A. Grasselli, Samuel Mather, I. B. Miller.

ENTERTAINMENT.—F. B. Richards, *Chairman;* H. W. Lash, B. D. Quarrie, W. Rattle, Jr., Willard Sawyer, R. B. Sheridan.

PROGRAM AND PRINTING.—C. B. Murray, *Chairman;* L. E. Dunham, F. A. Emmerton, G. F. Knapp, A. W. Smith, O. Textor.

GENERAL RECEPTION.—R. R. Abbott, M. A. Ammon, H. A. Barren, G. Bartol, H. H. Brown, W. N. Crafts, J. H. Cremer, B. Crowell, D. T. Croxton, S. W. Croxton, H. G. Dalton, L. E. Dunham, F. A. Emmerton, C. H. Fulton, C. A. Grasselli, H. C. Hale, L. E. Holden, R. F. Jopling, H. L. Kaufman, H. R. Kimmell, G. F. Knapp, W. Koehler, H. M. Lane, H. W. Lash, I. P. Lihme, A. G. McKee, M. McMurray, Samuel Mather, W. G. Mather, E. P. Merrill, H. G. Merry, L. B. Miller, C. B. Murray, E. W. Oglebay, F. J. Peck, T. E. Pierce, B. D. Quarrie, W. J. Rattle, F. B. Richards, L. A. Roby, H. E. Seidl, J. H. Sheadle, A. W. Smith, O. Textor, F. H. Treat, F. R. Van Horn, O. M. Weichsel, S. T. Wellman, R. Zeising.

IRON AND STEEL DIVISION.

Charles Kirchhoff, *Chairman;* Charles F. Rand, *Vice-Chairman;* Bradley Stoughton, *Secretary;* John Birkinbine, James Gayley, Henry D. Hibbard, Henry M. Howe, Charles K. Leith, Robert W. Hunt, Julian Kennedy, Richard Moldenke, Joseph W. Richards, Albert Sauveur, F. W. C. Schniewind, A. A. Stevenson, Felix A. Vogel, Leonard Waldo, William R. Webster.

INSTITUTE HEADQUARTERS, Hotel Statler.

On Monday evening, Oct. 28, 1912, the visiting members and guests were informally received by the Local Committee at the Headquarters of the Institute at the Hotel Statler. The evening was very pleasantly spent in renewing old acquaintances and friendships and in forming new ones.

The first session, held Tuesday morning, Oct. 29, 1912, in the impressive Assembly-Room of the hotel, was called to order by David T. Croxton, Chairman of the Local Committee. Mr. Croxton introduced the Hon. Newton D. Baker, Mayor of Cleveland, who extended a cordial welcome to the members and guests present. James F. Kemp, President of the Institute, who subsequently presided at the meeting, responded in behalf of the Institute.

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The following papers were presented in oral abstract by the authors:

* Recent Developments in the Inspection of Steel Rails, by Robert W. Hunt, Chicago, Ill.

Discussion by Henry D. Hibbard, Frederick Laist, Elwood Haynes, William H. Blauvelt, R. H. Sweetser, and reply by Mr. Hunt.

* Notes on Titanium and on the Cleansing Effect of Titanium on Cast-Iron, by Bradley Stoughton, New York, N. Y.

Discussion by J. E. Johnson, Jr., James F. Kemp, and reply by Mr. Stoughton.

† The Iron-Ores of the South Range of the Cuyuna District, Minnesota, by Carl Zapffe and W. A. Barrows, Jr., Brainerd, Minn.

Discussion by D. T. Croxton.[1]

* The Action of Various Commercial Carbonizing-Materials, by Robert R. Abbott, Cleveland, Ohio.

Discussion by Henry D. Hibbard and reply by Mr. Abbott.

* Measurements and Relations of Hardness and Depth of Carbonization in Case-Hardened Steel, by Mark A. Ammon, Cleveland, Ohio.

Discussion by J. E. Johnson, Jr., Bradley Stoughton, and reply by Mr. Ammon.

(A unanimous vote of thanks was passed to the Peerless Motor Car Co., for their liberal policy in placing at the disposal of the Institute the information presented by Messrs. Abbott and Ammon in their excellent papers.)

‡ Wittorff's Iron-Carbon Equilibrium Diagram, by Bradley Stoughton, New York, N. Y.

* Notes on Ruff's Carbon-Iron Equilibrium Diagram, by Henry M. Howe, New York, N. Y. (Presented by Bradley Stoughton.)

Discussion by J. E. Johnson, Jr., Henry D. Hibbard, Bradley Stoughton, R. H. Sweetser, James F. Kemp, and reply by Mr. Stoughton.

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* Papers distributed in pamphlet form at the meeting.
† Papers distributed in mimeograph form at the meeting.
‡ Papers in manuscript form available for consultation.
[1] Not furnished for publication.

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The second session, on Tuesday afternoon, held in the same place, was called to order by President Kemp.

The following papers were presented in oral abstract by the authors:

* Blowing-In a Blast-Furnace, by Ralph H. Sweetser, Columbus, Ohio.

Discussion by J. E. Johnson, Jr.

* Alloys of Cobalt with Chromium and Other Metals, by Elwood Haynes, Kokomo, Ind.

Discussion by P. N. Moore, Carl Zapffe, R. R. Abbott, S. T. Wellman, J. E. Johnson, Jr., and reply by Mr. Haynes.

* The Manufacture of Coke, by William H. Blauvelt, Syracuse, N. Y.

* By-Product Coke, by Charles W. Andrews, Duluth, Minn. (Presented by Dr. Schniewind.)

* The Manufacture of Coke, by F. E. Lucas, Sydney, N. S. (Presented by William H. Blauvelt.)

The above three papers on the manufacture of coke were discussed by Joseph E. Thropp, Jr., James F. Kemp, E. V. d'Invilliers, Dr. F. Schniewind, E. W. Parker, Dr. R. Moldenke (communication presented by Mr. Stoughton), Bradley Stoughton, H. H. Stoek, Charles H. Fulton, Ralph H. Sweetser, and replies by Mr. Blauvelt and Dr. Schniewind.

* The Concentration of Iron-Ores, by N. V. Hansell, New York, N. Y.

Discussion by E. G. Spilsbury, P. N. Moore, J. E. Johnson, Jr., E. V. d'Invilliers.

* The Effect of Alumina in Blast-Furnace Slags, by J. E. Johnson, Jr., Ashland, Wis.

Discussion by Ralph H. Sweetser, Charles H. Fulton, James F. Kemp, and reply by Mr. Johnson.

The third session, Wednesday morning, Oct. 30, held in the same place, was devoted to a discussion of the business affairs of the Institute. President Kemp called the meeting to order and prefaced the proceedings with a brief history of the Institute management to date, calling attention to the fact that the Institute now being an incorporated body of the State of New

* Papers distributed in pamphlet form at the meeting.

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York, by its charter can take no legal action at meetings held outside of the State of New York. The present session therefore could only be devoted to discussion.

Charles F. Rand, in behalf of the Special Committee appointed by the Board of Directors and the Council (Joseph W. Richards, *Chairman;* Charles Kirchhoff, and Charles F. Rand), presented printed copies of the proposed new Constitution and By-Laws, which were distributed among the members. The various articles were taken up one by one, special attention being given to Art. IV., which relates to the establishment of a class of "Fellows" in the Institute. This Article was submitted more for discussion than as a recommendation. Discussion from the floor was held by C. R. Corning, Charles Kirchhoff, J. W. Malcolmson, P. N. Moore, H. H. Stoek, Robert W. Hunt, J. E. Johnson, Jr., H. D. Hibbard, E. V. d'Invilliers, H. F. Bain, E. W. Parker, Bradley Stoughton, A. R. Ledoux, William Kelly, E. C. Spilsbury, Carl Zapffe, S. L. Goodale, E. H. Benjamin, Joseph Struthers.

The Article as drawn granted power to the body of Fellows to express an opinion as such with regard to "pending or proposed legislation affecting the mining or metallurgical industries." A vote was called to ascertain the sense of those present concerning this question as put. The result was 37 against and 17 in favor. The broad question of establishing a class of Fellows was then presented and voted upon, with the result of 27 against and 20 in favor.

The proposed By-Laws were briefly presented and a special discussion was held on By-Law III., referring to the duties of the Secretary; especially the section which required that the Secretary should devote his entire time to the affairs of the Institute. Discussion from the floor was held by J. E. Johnson, Jr., Charles F. Rand, A. R. Ledoux, Charles Kirchhoff, E. G. Spilsbury, S. L. Goodale, James F. Kemp, and S. W. Malcolmson. On request, an expression of opinion was asked as to whether the restriction should remain, namely, that the entire time of the Secretary should be devoted to Institute affairs; the vote showed 13 in favor of the restriction and 30 against it.

C. R. Corning presented arguments in support of the proposed amendments to the Constitution and By-Laws presented by himself and George C. Stone at the adjourned Annual Busi

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ness Meeting of the Institute, Oct. 7, 1912, and subsequently sent in printed form to the membership at large in circular No. 8, Oct. 15, 1912.

The fourth and concluding session was held Thursday morning at the Case School of Applied Science in the large lecture-room of the Electricity Building. Professor Kemp presided. The following papers were presented in oral abstract:

† The Effect of High Carbon on the Quality of Charcoal-Iron, by J. E. Johnson, Jr., Ashland, Wis.

Discussion by Bradley Stoughton and by letter from Henry M. Howe and Richard Moldenke.

* The Constitution and Melting-Points of a Series of Copper-Slags, by Charles H. Fulton, Cleveland, Ohio.

Discussion by Frederick Laist and Henry D. Hibbard.[2]

* Chemistry of the Reduction Processes in Use at Anaconda, Mont., by Frederick Laist, Anaconda, Mont.

* Note on the Case-Hardening of Special Steels, by Albert Sauveur and G. A. Reinhardt, Cambridge, Mass. (Presented by Mr. Reinhardt.)

Discussion presented in writing by R. R. Abbott.

A discussion of the paper of R. W. Raymond on Our National Resources and Our Federal Government, by George Otis Smith, Washington, D. C. (Presented by E. W. Parker.)

C. R. Corning presented orally a few additional remarks concerning the proposed amendments to the Constitution and By-Laws, discussed by him at the third session.

The proceedings concluded with a cordial vote of thanks to the Local Committee for the complete and delightful manner in which the comfort and enjoyment of the members had been looked after; also to the Case School of Applied Science for placing at disposal of the Institute for the final session the lecture-room in the Electricity Building.

In addition to the papers already noted, the following were read by title for publication:

* Surveying and Sampling Diamond-Drill Holes, by E. E. White, Ishpeming, Mich.

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* Papers distributed in pamphlet form at the meeting.

† Papers distributed in mimeograph form at the meeting.

[2] Not furnished for publication.

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† The Utility of Efficiency-Records in the Manufacture of Iron, by John J. Porter, Staunton, Va.

* Fuel-Efficiency of the Cupola-Furnace, by John J. Porter, Staunton, Va.

* The Microstructure of Iron and Steel, by William Campbell, New York, N. Y.

* Recent Developments in Open-Hearth Steel-Practice, by N. E. Maccallum, Phœnixville, Pa.

† The Influence of Divorcing Annealing on the Mechanical Properties of Low-Carbon Steel, by Henry M. Howe and Arthur G. Levy, New York, N. Y.

† Methods of Preparing Basic Open-Hearth Steel for Castings, by H. F. Miller, Jr., Verona, Pa.

† Electric Heating and the Removal of Phosphorus from Iron, by A. E. Greene, Chicago, Ill.

* A Titaniferous Iron-Ore Deposit in Boulder County, Colo., by E. P. Jennings, Salt Lake City, Utah.

† The Function of Slag in Electric Steel-Refining, by R. Amberg, East Orange, N. J.

* The Ta-yeh Iron-Ore Deposits, Hu-pei Province, China, by C. M. Weld, Low Moor, Va.

* Clinton Iron-Ore Deposits in Kentucky and Tennessee, by S. Whinery, New York, N. Y.

* Melting Iron in the Cupola-Furnace, by R. Moldenke, Watchung, N. J.

* Centrifugal Machines for Ore-Grading and Ore-Concentrating, by Godfrey T. Vivian, Cornwall, England.

* The St. Helens Mining-District, by Horace V. Winchell, Minneapolis, Minn.

* Fires in Metalliferous Mines, by George J. Young, Reno, Nev.

* Present Conditions of Mining in the District of Vladivostok, Siberia, by Albert F. J. Bordeaux, Thonon-les-Bains, France.

* The Sampling of Gold-Bullion, by Frederic P. Dewey, Washington, D. C.

* An Experience in the Use of Water-Power, by C. M. Myrick, San Francisco, Cal.

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* Papers distributed in pamphlet form at the meeting.

† Papers distributed in mimeograph form at the meeting.

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* Our National Resources and Our Federal Government, by R. W. Raymond, New York, N. Y.

* The Wood Flotation Process, by Henry E. Wood, Denver, Colo.

* The Occurrence of Gold in the Eocene Deposits of Texas, by E. T. Dumble, Houston, Texas.

* A Graphic Solution of D'Arcy's Formula for the Transmission of Compressed Air in Pipes, by Nathaniel Herz, Lead, S. D.

* The Alundum Extraction-Thimble Used in the Determination of Copper, by L. W. Bahney, New Haven, Conn.

* Notes on the Metallography of Alloys, by William Campbell, New York, N. Y.

‡ The Ultimate Source of Metals, by Blamey Stevens, Temascaltepec, Mexico.

* Notes on Bag-Filtration Plants, by Anton Eilers, New York, N. Y.

* The Development of the Parkes Process in the United States, by Ernst F. Eurich, New York, N. Y.

* The Sulphatizing-Roasting of Copper-Ores and Concentrates, by Utley Wedge, Ardmore, Pa.

‡ Development of the American Water-Jacket Lead Blast-Furnace, by R. C. Canby, Wallingford, Conn.

‡ Heat-Losses in Furnaces, by F. A. J. Fitzgerald, Niagara Falls, N. Y.

* The Development of the Reverberatory Furnace for Smelting Copper-Ores, by E. P. Mathewson, Anaconda, Mont.

‡ A Trip Through Northern Korea, by H. W. Turner, London, England.

The Methods of the United States Steel Corporation for the Commercial Sampling and Analysis of Pig-Iron, by J. M. Camp.

‡ New Type of Blast-Furnace Construction, by J. E. Johnson, Jr., Ashland, Wis.

‡ Discussion of F. G. Cottrell's paper, Electrical Fume-Precipitation, by Richard Peters, Jr., Barker, Pa.[3]

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* Papers distributed in pamphlet form at the meeting.

‡ Papers in manuscript form available for consultation.

[3] *Trans.*, xliii., 760 (1912).

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Excursions and Entertainments.

On Tuesday evening, Oct. 29, an informal smoker in the large Assembly-Room of the Statler Hotel was tendered to the members and guests of the Institute party, and through the courtesy of Rogers, Brown & Co., a remarkably vivid motion-picture portrayal was presented entitled "From Mine to Molder," beginning with the mining of the ore and showing the methods and complete processes until the finished material is produced. A running explanation of the various steps of the process was given by Henry B. B. Yergason, a representative of the company. The smoker concluded with an interesting illustrated address by Prof. James F. Kemp and an informal collation.

On Tuesday afternoon, Oct. 29, during the technical session, the visiting ladies were given an automobile ride through the delightful boulevards of the city and later entertained with a tea at the Country Club.

On Wednesday afternoon, Oct. 30, the party was taken by a special train, provided through the kindness of the American Steel & Wire Co., to the Central and Newburg plants of that company. Mr. B. D. Quarrie, the General Superintendent, and a corps of assistants piloted the party in groups through both plants. The return to headquarters was made by special trolley-cars.

On Wednesday evening a banquet to more than a hundred was held in the Assembly-Room of the hotel, which was tastefully decorated with flowers. Mr. D. T. Croxton, the genial toast-master, contributed largely to the enjoyment of the occasion. Informal addresses were made by Capt. Robert W. Hunt, W. G. Mather, E. W. Parker, E. H. Benjamin, Philip N. Moore, Prof. James F. Kemp, and Charles Kirchhoff. Telegrams of greeting were sent to "Uncle" John Fritz, of Bethlehem, Pa., and Dr. R. W. Raymond, at Washington, Conn.

On Thursday afternoon, Nov. 1, the members and guests were taken by automobiles to the extensive plant of the National Carbon Co., where, through the courtesy of Mr. Henry A. Barren, the General Superintendent, and a number of active assistants, the various departments were visited and inspected.

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Through the kindness of E. W. Kronbach, of the National Carbon Co., the following brief description of the interesting visit is here given:

The itinerary started with an inspection of the company's library, which contains practically all the up-to-date technical literature, magazines and books, which could in any way help its employees. The arc-lamp testing-laboratory was next visited, where several hundred arc-lamps, representing nearly every type of lamp in commercial use, were under test to prove the carbon electrodes. In the room above hundreds of dry cells were being tested in every conceivable way. A large number of these cells were tested merely to check the uniformity of the daily output, while others were experimental cells, made up by the Research Department, to improve, if possible, the present product.

The factory was then visited and the manufacture of dry cells watched through the various processes of shaping and soldering the zinc cans, lining them, inserting the electrode, tamping the mix, soldering on the binding-post, and finally pouring in the seal. The several careful testing-methods to which each cell is subjected to insure that only perfect cells should leave the factory were carefully explained and noted.

The route then lay through the mixing- and grinding-rooms, where the various ingredients for the lighting-carbons, electrodes, brushes, and cells are thoroughly ground and mixed. From here the forcing-room, where all the arc-lamp carbons are forced through dies, was visited. In the upper end of the building the brushes and the dry-cell electrodes are molded under enormous hydraulic pressure. The molding, forcing and stamping of the many carbon specialties, such as contacts, plungers, disks, and diaphragms of all sizes and shapes, created a great deal of interest. Next was visited the furnace-room, containing furnaces to bake the carbon products and impart the necessary electrical and physical properties.

The balance of the work seen was more of a mechanical nature, and consisted in cleaning from the adhering packing-materials the carbon products, after coming from the furnaces. The lighting-carbons are then put through the sorting, gauging and coring processes, while the brushes are cut, planed, beveled and copper-coated, according to the demands of the trade. The various products are then wrapped, packed and shipped.

In the last place visited—the brush-testing laboratory—are machines, both electrical and mechanical, to test the brushes thoroughly before shipping. The following tests are made: specific resistance, strength, hardness, density, friction-tests at various speeds, contact-resistance tests at different brush-pressures and current-densities.

A trip to the ore-docks of M. A. Hanna & Co. followed, and the party viewed with awe the impressive motions of the Hewlett unloader, which was discharging with its massive 17-ton bucket the cargo of iron-ore from the S. S. *David B. Meacham.*

Special mention should be made of the attractive and interesting souvenir program prepared by the Program Committee under Mr. C. B. Murray, *Chairman.* This small volume, cloth-bound in attractive green, contains not only the data of officers,

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committees, titles and authors of papers, and other matters of routine importance; but also a brief illustrative description of the principal buildings, parks, and other public institutions for which Cleveland is noted,

The following list, doubtless incomplete, comprises the names of members and guests enrolled at Headquarters:

Members and Guests in Attendance at the Sessions and Excursions.

Abbott, Robert R., Cleveland, Ohio.
Ammon, Mark A., Cleveland, Ohio.
Applegate, J. S., Youngstown, Ohio.
Arluck, A., Cleveland, Ohio.
Backert, A. O., Cleveland, Ohio.
Badger, A. C., Cleveland, Ohio.
Bain, H. Foster, San Francisco, Cal.
Barren, Henry A., Cleveland, Ohio.
Barrows, W. H., Jr., Duluth, Minn.
Benjamin, E. H., San Francisco, Cal.
Bever, C. A., Cleveland, Ohio.
Blauvelt, W. H., Syracuse, N. Y.
Bowler, R. P., New York, N. Y.
Braid, Arthur I., New York, N. Y.
Braman, H. T., Youngstown, Ohio.
Brayer, M. S., Sharon, Pa.
Byrne, J. H., Cleveland, Ohio.
Callender, L. W., Cleveland, Ohio.
Carey, C. S., Cleveland, Ohio.
Chute, H. O., New York, N. Y.
Clagett, Thomas H., Bluefield, W. Va.
Coe, W. W., Roanoke, Va.
Collier, L. A., Cleveland, Ohio.
Collord, George L., Sharon, Pa.
Cook, Edgar S., Pottstown, Pa.
Corning, C. R., New York, N. Y.
Cox, Jennings S., Jr., Santiago de Cuba.
Crafts, W. N., Cleveland, Ohio.
Crowell, Benedict, Cleveland, Ohio.
Croxton, D. T., Cleveland, Ohio.
Croxton, Mrs. D. T., Cleveland, Ohio.
Croxton, S. W., Cleveland, Ohio.
Dalton, H. G., Cleveland, Ohio.
Danforth, A. E., Cleveland, Ohio.
D'Invilliers, E. V., Philadelphia, Pa.
Dunham, L. E., Cleveland, Ohio.
Dunham, Mrs. L. E., Cleveland, Ohio.
Emmerton, F. A., Cleveland, Ohio.
Emmerton, Mrs. F. A., Cleveland, Ohio.
Eustis, Augustus H., Boston, Mass.
Eustis, W. E. C., Boston, Mass.
Findley, A. I., New York, N. Y.
Ford, E. L., Youngstown, Ohio.
Fowler, D. D., Cleveland, Ohio.
Fruehauf, P. A., Cleveland, Ohio.
Fulton, Charles H., Cleveland, Ohio.
Fulton, H. F., Cleveland, Ohio.
Gayley, James, New York, N. Y.
Goodale, Stephen L., Pittsburg, Pa.
Gresham, A. L., New York, N. Y.
Griswold, C. L., Pittsburg, Pa.
Hale, H. C., Cleveland, Ohio.
Hamilton, J. H., New York, N. Y.
Hansell, N. V., New York, N. Y.
Harrison, H. T., Cleveland, Ohio.
Haynes, Elwood, Kokomo, Ind.
Heston, R. P., Toledo, Ohio.
Hibbard, Henry D., Plainfield, N. J.
Hindshaw, Henry H., Alpena, Mich.
Hoyt, W. H., Cleveland, Ohio.
Humphrey, George S., New York, N. Y.
Humphrey, Mrs. G. S., New York, N. Y.
Hunt, Robert W., Chicago, Ill.
Ireland, J. D., Duluth, Minn.
Ives, L. E., Cleveland, Ohio.
Jewitt, F. G., Minneapolis, Minn.
Johnson, J. E., Jr., Ashland, Wis.
Johnson, Mrs. J. E., Jr., Ashland, Wis.
Johnson, Nason, Alpena, Mich.
Kelly, William, Vulcan, Mich.
Kelly, Mrs. William, Vulcan, Mich.
Kemp, James F., New York, N. Y.
King, Paul H., Philadelphia, Pa.
Kirchhoff, Charles, New York, N. Y.
Kirchhoff, Mrs. Chas., New York, N. Y.
Knapp, G. F., Cleveland, Ohio.
Koehler, William, Cleveland, Ohio.
La Follette, H. M., La Follette, Tenn.
Laist, Frederick, Anaconda, Mont.
Lake, E. F., Cleveland, Ohio.
Lash, Horace W., Cleveland, Ohio.
Ledoux, A. R., New York, N. Y.

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Lihme, I. P., Cleveland, Ohio.
Livingston, J. B., Cleveland, Ohio.
Longyear, E. J., Minneapolis, Minn.
Lyon, D. A., Pittsburg, Pa.
McKee, A. G., Cleveland, Ohio.
Macon, W. W., Brooklyn, N. Y.
Malcolmson, James W., Kansas City, Mo.
Marshall, S. W., Johnstown, Pa.
Marting, H. A., Ironton, Ohio.
Mather, Samuel, Cleveland, Ohio.
Mather, W. G., Cleveland, Ohio.
Menke, Paul O., Sharon, Pa.
Merry, H. G., Cleveland, Ohio.
Miller, L. B., Cleveland, Ohio.
Moore, Philip N., St. Louis, Mo.
Motter, W. D. B., Jr., Trenton, Ont.
Murray, C. B., Cleveland, Ohio.
Murray, Mrs. C. B., Cleveland, Ohio.
Parker, Edward W., Washington, D. C.
Parrock, H. P., Buffalo, N. Y.
Pecanka, William, Cleveland, Ohio.
Pelton, J. S., New York, N. Y.
Pfeiffer, George W., Santiago de Cuba.
Pierce, C. B., Cleveland, Ohio.
Pitkin, S. H., Akron, Ohio.
Pitkin, Mrs. S. H., Akron, Ohio.
Playter, Franklin, Boston, Mass.
Price, J. D., New Straitsville, Ohio.
Price, John M., Cleveland, Ohio.
Rand, Charles F., New York, N. Y.
Rand, Mrs. Chas. F., New York, N. Y.
Ranney, W. B., Lakewood, Ohio.
Rattle, William, Jr., Cleveland, Ohio.
Read, J. B., Cleveland, Ohio.
Reinhardt, G. A., Cambridge, Mass.
Reitz, J. C., West Park, Ohio.
Richards, F. B., Cleveland, Ohio.
Robinson, C. Snelling, Youngstown, Ohio
Roeber, E. F., New York, N. Y.
Sawyer, Willard N., Cleveland, Ohio.
Schoefer, C. A., Cleveland, Ohio.
Schumacher, Dr. W., Berlin, Germany.
Schniewind, Dr. F., Englewood, N. J.
Sheridan, R. B., Cleveland, Ohio.
Sherrerd, J. M., Easton, Pa.
Stillwagon, S. C., Cleveland, Ohio.
Smith, Albert W., Cleveland, Ohio.
Spilsbury, E. Gybbon, New York, N. Y.
Stay, T. D., Cleveland, Ohio.
Stebbins, H. S., Cleveland, Ohio.
Stoek, H. H., Urbana, Ill.
Stoughton, Bradley, New York, N. Y.
Stoughton, Mrs. Bradley, New York, N. Y.
Struthers, Joseph, New York, N. Y.
Sweetser, R. H., Columbus, Ohio.
Sweetser, Mrs. R. H., Columbus, Ohio.
Taylor, Knox, High Bridge, N. J.
Taylor, Mrs. Knox, High Bridge, N. J.
Textor, Oscar, Cleveland, Ohio.
Textor, R. B., Cleveland, Ohio.
Thropp, J. E., Jr., Indiana Harbor, Ind.
Thropp, Mrs. J. E., Jr., Indiana Harbor.
Treat, F. H., Cleveland, Ohio.
Vallat, B. W., Ironwood, Mich.
Van Horn, Frank R., Cleveland, Ohio.
Vogel, Felix A., New York, N. Y.
Volk, Karl E., West Park, Ohio.
Walther, Fred. F., Cleveland, Ohio.
Weld, C. M., Low Moor, Va.
Wellman, S. T., Cleveland, Ohio.
Wendel, Edmund, Cleveland, Ohio.
Whelan, M. T., Cleveland, Ohio.
Wood, C. L., Cleveland, Ohio.
Wood, Walter, Philadelphia, Pa.
Yergason, H. B. B., Cincinnati, Ohio.
Yoeum, G. A., Cleveland, Ohio.
Zapffe, Carl, Brainerd, Minn.
Zay, Jeffries, Cleveland, Ohio.

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PAPERS.

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The Iron-Ores of the South Range of the Cuyuna District, Minnesota.

BY CARL ZAPFFE AND W. A. BARROWS, JR., BRAINERD, MINN.

(Cleveland Meeting, October, 1912.)

Introduction.

THE Cuyuna iron-ore district lies in Aitkin, Crow Wing, and Morrison counties of the State of Minnesota. The developed part of the district lies wholly within the southerly portion of Crow Wing county, crossing it in a NE–SW. direction, practically parallel to the course of the Mississippi river. The Brainerd-Duluth branch of the Northern Pacific railway is also practically parallel to this course and, in addition, divides the Cuyuna district into two geographic divisions. That portion of the district lying north of the railway is known locally as the North Range, and, correspondingly, that portion lying south of the railway is known locally as the South Range. There appear to be no geological differences between these two subdivisions.

The commercially important portion of the South Range is entirely in Crow Wing county and embraces an area about 30 miles long and about 2 miles wide. The purpose of the present paper is the consideration of the ores of this portion of the South Range. The information is obtained entirely from drill-hole records, for there are no rocks outcropping in the district.

Geology.

The principal rock-formation consists of metamorphosed sedimentaries interbedded with large amounts of basic volcanic flows and tuffs, all presumably of Upper Huronian age. This formation is intruded by acid and basic plutonic rocks of Keweenawan age. There are also overlying remnants of an acid amydaloidal surface-flow, presumably of Keweenawan age, and a conglomerate of Cretaceous age, but as both are unimportant, they will not be further discussed.

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The sediments consist principally of different colored and schistose slates, interbedded with small quantities of limestone, quartzite, graywacke, and the various iron-bearing rocks. The latter seem to occupy a more or less definite horizon in the Upper Huronian group, and are known as the Deerwood member of the Virginia slate formation.[1]

The plutonic rocks are gabbros, diabases, quartz-diorites, and granite. Drilling has not yet definitely outlined the forms in which these rocks have intruded the Cuyuna formations, but they seem to be in the form of small dikes and bosses. Granite is known only in outlying areas.

The volcanic and sedimentary rocks of the Cuyuna district have been folded into numerous similar folds parallel in strike and of low pitch, so low in places that the crest-line of some folds appears to be measured by miles in length. The deformation has produced closed folds of a prevailingly steep dip, both to the NW. and SE., and the remaining limbs, not removed by erosion, show many minor flexures, especially in the nature of drag-folds along the strike. Nothing is known that would indicate definitely the nature of the underlying rocks, and meager remnants of a younger conglomerate and an extrusive lie uncomformably above. The folded rocks are intruded by the plutonic rocks. There has been considerable fracturing and minute faulting, and much recementing by quartz. Erosion has removed great thicknesses of rocks and has exposed the iron-bearing member, making conditions favorable for oxidation and the development of the ores at exposed surfaces.

Iron-Bearing Member.

The iron-bearing member consists of ferruginous slates, ferruginous cherts, amphibole-magnetite rocks (locally called "magnetic slates"), cherty iron carbonates, and iron-ores. The ferruginous slates are soft, the ores are both soft and hard, and banded and brecciated, the ferruginous cherts are dense but generally brittle, and the "magnetic slates" and the cherty iron carbonates are dense and fairly hard. The

[1] "Virginia slate" is the name given to this Cuyuna Upper Huronian formation by the U. S. Geological Survey, *Monograph LII.*, being correlated by Leith and Van Hise with the Virginia slate formation of the Mesabi district.

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magnetic qualities of the "magnetic slates" enable tracing the concealed iron-bearing member in spite of the lack of rock outcrops.

Ore-Deposits.

On the whole the ore-bodies are lens-shaped, but assume a variety of forms within short distances. They may be broad or narrow, single or double, and in part parallel, separated by lean iron-bearing material or barren rock, all leading to complicated conditions and outlines. They are steeply inclined, dipping either to the NW. or SE. at from 70° to 90° from the horizontal. Because of the overlapping of nearby lenses of ore, some ore-deposits, considering material containing more than 40 per cent. of iron as constituting an ore, appear to be a mile or more in length, but the length at the surface of the average commercially valuable deposit is probably nearer three-quarters of a mile. The greatest known width of an ore-body at the surface is about 300 ft., but the more common maximum width is nearer 125 feet.

Because such a very large percentage of the drilling has been done with shallow inclined holes, very little is known about the downward continuation of these ore-bodies. Concentration has been encountered to a depth of 300 ft. The ore-bodies occur in basins or troughs, formed by folding, by igneous intrusives, or by both. These basins or troughs pitch either to the NE. or the SW., whereas the ore-bodies dip to the SE. or to the NW. It is easy to conceive of an inclined hole either under-cutting or over-cutting such a pitching ore-body, and thereby indicating that the ore is absent entirely or possibly only of shallow depth, when, as a matter of fact, it is more likely that the ore probably continues down the pitch and has a greater depth than any drilling in the vicinity may show. Such structural relations are not only favorable for the development of ore to considerable depth, but also for the development of the richer deposits. It remains for actual underground operations to verify these facts for the Cuyuna district; however, the general geological structure is most favorable. It is due to the present state of developments, therefore, that the richest ores of the South Range deposits are shown occupying approximately only the upper 100 ft. of the deposit.

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The ore-bodies are incased in varying amounts of lean iron-bearing formation, which usually grades into the barren phases of the Virginia slate formation. Any of these rocks, and the intrusives, may form the hanging-wall or foot-wall of a deposit. A valuable feature of the ore-bodies is that they are strikingly free of large quantities of interbedded lean material.

It is impossible to state to what extent folding and faulting have distorted or dislocated ore-bodies, but it is to be expected that both will be found to have been effective; likewise intrusive dikes, which are known in some districts to have lengthened the lives of many mines by having produced favorable conditions in places for the enrichment and development of ore-deposits which were in no manner suggested in the earlier exploratory work.

Twenty good ore-deposits have already been well outlined by drilling, and many more have been at least indicated and are deserving of further exploration. As most of the deposits were not explored for immediate operations, and as the drilling has been done by means of inclined holes, there are, therefore, two factors seriously affecting the figures of developed tonnages, namely, the lack of complete exploration, and the lack of knowledge of depth of ores. The better deposits, as now known, generally cross from two to four forties and contain at least 1,000,000 tons of material, all of which contains 50 per cent. or more of iron. The average iron-content is variable, and for each deposit can be regulated by selection.

The minute interbanding of the ores with leaner phases causes the chemical composition to show rapid alternation across the strike. The angle-hole drilling on these steeply-inclined beds, therefore, gives a truer conception of the composition of the deposit than vertical drilling.

The Ores.

The ores are mostly the hydrous and siliceous limonites of red, brown, and black color, the brown color predominating. When not very siliceous the ores are earthy in appearance. Hematite is rare, and though parts of the iron-bearing member are magnetic, no magnetite-deposits exist. The different kinds of ores occur interlayered.

The ores have been brecciated considerably, but are gener-

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ally recemented by secondary iron. Some ores are granular, but not so fine as to make them objectionable in the furnaces. Drilling has not clearly indicated the true average textural character of the ores, but it seems that the deposits as a whole are relatively soft and brittle, due to brecciation, but the breccia fragments themselves are hard. There are no hard ores like the Vermilion and Marquette hard ores, nor soft ores like the Mesabi ores. The best counterparts for Cuyuna ores can be found in the Iron River and Crystal Falls districts of Michigan.

The density of the average Cuyuna ores is about 4.10. The pore-space of the harder ores varies from 9 to 20 per cent. by volume, and for the softer ores ranges up to 36 per cent.

The harder banded ores average 10 cu. ft. per ton, the brecciated hard ores range from 10 to 10.8, the soft ores average 11.5, and the lean soft ores average 12.6 cu. ft. per ton. These figures have been determined by laboratory-experiments, and will have to suffice until determinations can be made on deposits under operation. It is impossible now, in estimating, to separate the various kinds of ores, so 10 and 12 cu. ft. per ton are the preferable factors for the better and the leaner grades respectively.

An estimate of the tonnage developed by the drilling on only the explored forties shows that at least 20,000,000 tons of ore, none of which contains less than 50 per cent. of iron, as shown on the exploration-sheets, have actually been developed. By making greater and plausible assumptions in dimensions, governed by the geological conditions, this figure is increased to about 50,000,000 tons. The total amount of iron-bearing formation indicated by this same drilling, and containing more than 40 per cent. of iron, ranges between 50,000,000 and 110,000,000 tons. No attempt has been made to separate the 60-per cent. ore, as only a small amount has been developed up to the present time.

An estimate of the average iron-content, governed by the tonnage figures of the different deposits, of all that material shown on the exploration-sheets containing 50 per cent. or more of iron, shows the average iron-content to be 55.0 per cent. (dried at 212° F.). A similar calculation of only that material analyzing 55 per cent. or more in iron gives an average of 57.3 per cent. As explorations continued, both the

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tonnages of 50-per cent. ores and the calculated average iron-content of the same ever increased, and this can be expected to continue, inasmuch as the explorations consist of but a mere scratch compared to the probable total possibilities.

Of the total tonnage of ores exceeding 40 per cent. of iron, 75 per cent. contains more than 45 per cent. of iron; 45 per cent. more than 50 per cent. of iron, and 20 per cent. more than 55 per cent. of iron.

Complete ore-analyses are available for only parts of some deposits. In Table I. are listed in consecutive order of increasing percentages of the iron-content some of the available partial analyses which contain determinations for at least three elements. The analyses represent the chemical composition of only portions of deposits, but are properly corrected for the tonnage represented.

TABLE I.—*Chemical Compositions of South Range Ores, Cuyuna District, Minnesota.*

Fe (Dry 212°).	P.	Mn.	SiO_2.	Al_2O_3.	CaO.	MgO.	S.	Loss on Ignition
Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	P'r Cent.
48.4	0.599	1.223	14.69	1.15	.....	.....	.....	.....
50.1	0.341	0.545	17.70	0.73	.....	.....	.....	.....
50.9	0.387	0.357	.....	.....	.....	.....	.....	.....
50.9	0.335	0.63	.....	.....	.....	.....	.....	.....
51.2	0.459	0.765	15.41	1.98	.....	.....	0.039	.....
53.7	0.703	0.339	8.99	3.20	.....	.....	0.041	.....
53.8	0.396	0.487	9.26	2.06	0.104	0.109	0.043	8.95
54.1	0.425	0.299	.....	.....	.....	.....	.....	.....
54.4	0.315	1.38	6.62	1.87	0.256	0.150	0.035	9.22
54.5	0.301	0.95	.....	.....	.....	.....	.....	.....
55.2	0.389	0.265	10.32	1.85	.....	.....	0.067	5.75
55.4	0.451	0.835	8.42	1.95	0.34	0.141	0.018	7.14
55.7	0.535	0.383	5.49	2.93	0.28	0.146	0.011	9.95
56.3	0.366	0.99	7.80	1.90	0.157	0.135	0.018	6.43
57.1	0.311	0.235	10.89	1.02	0.133	0.059	.....	.....
57.2	0.339	.....	4.53	.....	.....	.....	.....	.....
58.1	0.324	0.357	9.35	0.996	0.168	0.055	.....	.....
60.6	0.272	0.28	.....	.....	.....	.....	.....	.....

It is evident that in the existing stage of development no serious consideration can be given to any one's approximate figures of average chemical constituents of the ores when determined by only a casual observation. Blue prints showing vertical holes in ore give a wrong impression of the conditions surrounding ore-deposits in the Cuyuna district. It is necessary that each ore-deposit be studied in detail, that the ore-

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analyses be carefully weighted according to tonnage figures, and that numerous deposits be represented.

Close inspection readily reveals the fact that the different minable deposits along the South Range vary only a trifle from each other in the percentages of their respective chemical constituents; therefore, an average chemical composition of South Range ores can be satisfactorily determined. Because of the similarity of deposits and because of the narrowness of the range of variation of the percentages of chemical constituents, such an average composition can well be considered as representative of South Range ores.

Table II. gives the chemical composition of what I consider the South Range type of minable and commercial iron-ore.

TABLE II.—*Type Composition of South Range Ore.*

	Per Cent.
Fe (Dried at 212°)	55.0
P	0.387
Mn	0.567
SiO_2	10.00
Al_2O_3	1.76
CaO	0.209
MgO	0.130
S	0.042
Loss on ignition	7.04

The following conditions pertain to individual deposits:

The average iron-content of any one deposit, calculated only from analyses of 50 per cent. or more, rarely exceeds 56.5 per cent. The iron-content varies directly with the silica-content. This variation as a whole is quite even for all material containing up to about 53 per cent. of iron, but for higher iron-values the silica decreases rapidly for every small increase in per cent. of iron. The average silica-content for ores containing more than 55 per cent. of iron seems to be less than 7 per cent. The highest content of iron obtained was 66 per cent.

Phosphorus has been determined in only the 50-per cent. ores. Rarely is an individual analysis under the Bessemer limit, and in only a few instances is it as high as 1.00 per cent. As a rule, the phosphorus-content of a deposit is strikingly uniform, usually averaging between 0.30 and 0.45 per cent.

The highest manganese analyses are only 2 per cent., and

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they occur in but a few cases. Like the phosphorus, manganese is quite uniform, and no one ore-deposit would average more than 0.75 per cent. of this element. This is in decided contrast with the North Range ores of the Cuyuna district, where large parts of many otherwise good ore-deposits are contaminated by excessive manganese.

A favorable feature of the deposits is the low alumina-content. It has already been stated that the deposits are remarkably clear of quantities of interlayered lean material. Inasmuch as the ores are associated with a slate formation, the low alumina-content is all the more noteworthy. Lean ores are as low in alumina as the richer ores. Individual analyses for alumina rarely reach 3 per cent., and the average for any deposit is 2 per cent. or less.

Calcium and magnesium are ever very low. Sulphur is always practically negligible, and no titanium has ever been reported.

Tests for loss on ignition range from 2.18 to 12.40 per cent. for all South Range ores, but in individual deposits the tests are all within a small range. This loss is due largely to the water combined in the hydrated minerals, and to a large extent to carbonates.

A figure of the moisture-content is mere speculation. A few early tests on a North Range ore-deposit gave 6.80, 10.40, and 14.20 per cent. for ores of different textures. South Range ores, it seems, will have a lower average moisture-content than the North Range ores. About 9 or 10 per cent. seems to be the correct amount to express this condition.

Comments on the Metallurgy of the Ores.

(By W. A. Barrows, jr., Brainerd, Minn.)

Attention is herein invited to what is known locally as the South Range of the Cuyuna iron-ore district, in Crow Wing county, Minn., which has until the past year been almost wholly neglected by the ore-consuming interests operating in the Lake Superior region. Explorations by means of churn- and diamond-drilling have been carried on in this district continuously for the past eight years. There is indicated by this work 110,000,000 tons of iron-ore, the average analysis of

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which is slightly below 50 per cent. of iron (dried at 212° F.), and includes no ore analyzing below 40 per cent. of iron. Included in this estimate are 50,000,000 tons having an average analysis of 54.60 per cent. of iron (at 212° F.), containing no ore sampled and analyzed showing less than 50 per cent. of iron.

It should be borne in mind that properties on the South Range only are included in these estimates. The North Range shows a tonnage greater than the South Range, contains several producing mines and is consequently better known. There is apparently no structural difference between ores from the North and South Ranges.

Active mining-operations on the South Range are being started at the Adams mine, 3 miles SW. of Deerwood, and at the Barrows mine, 3.5 miles SW. of Brainerd. The Adams mine-operation is conducted by Cuyler Adams, the pioneer of the Cuyuna district, and associates, and the Barrows mine by H. A. Hanna & Co., of Cleveland. Both of these properties are well equipped, the Adams having a concrete shaft. The main drift at the Barrows mine has just reached the ore-body and little ore has been hoisted. Both properties will enter the shipping-list in 1913.

The Adams and Barrows mines are 16 miles apart, and for this entire distance a continuous iron-formation has been shown by drilling. On the surface, two parallel lines or bands of magnetic attraction, nowhere more than 1.5 miles apart, can be traced over this 16-mile distance. This attraction continues in a SW. direction, and commercial iron-ore has been shown by drills 25 miles SW. of the Barrows mine.

A few years ago an exploring-shaft was sunk by Pickands, Mather & Co., of Cleveland, midway between the Adams and the Barrows mines (in Sec. 8, Twp. 45, Range 29), where drilling had shown an ore-body, and drifts and cross-cuts were run into the ore. This work confirmed the belief, gained from the study of drill-cores, that the Cuyuna ores were of excellent physical structure, the ore-body consisting of interbedded seams of hard and soft ore and being entirely free from the finer sandy material so common in Mesabi ores. This fine sandy material is the source of flue-dust in furnaces running on high percentages of Mesabi ores.

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In general appearance and physical structure Cuyuna ores closely resemble ores from the Marquette or Menominee Ranges. The iron-carrying ingredients of the South Range ores consist of a mixture of hematite, limonite, and iron carbonate. Taking the analysis of Mr. Zapffe's type composition of South Range ore as representative, and supposing a moisture-content of 10 per cent. (an amount confirmed so far by tests), gives an iron-content of 49.50 per cent. in natural condition. This analysis compares very favorably with the average analyses of the non-Bessemer ores shipped from the Mesabi Range during the past three years. It is probably not generally known, but is nevertheless true, that the average natural iron-content of all non-Bessemer ore shipped from the Mesabi Range in 1911 was 49.55 per cent.; in 1910, 49.65 per cent.; in 1909, 49.94 per cent.; and in 1902, 53.55 per cent. These figures show that the general average of all ore now classed as merchantable on the South Range is almost identical with the non-Bessemer Mesabi average for 1911, and that the average natural iron-content of Mesabi non-Bessemer ores for season's shipment has dropped 4 per cent. in 10 years.

The main advantage claimed for Cuyuna ores over ores from the Mesabi Range is that of better physical structure, their use insuring freedom from flue-dust troubles in smelting. A modern blast-furnace when driven to its capacity on all-Mesabi ore-mixture will probably entrain with the escaping gases not less than 5 per cent. of the ore charged, even when operating smoothly with ore of average Mesabi Range structure. During slips or periods of bad working, or when the finer-structure Mesabi ores are used, the dust-losses may greatly exceed 5 per cent. After considering carefully the difference in structure between Mesabi and Cuyuna ores, we conclude that ores from the Cuyuna Range will successfully resist the entraining effect of the escaping gases in the blast-furnace; moreover, when worked with Mesabi ores, they will greatly assist in reducing the blast-pressure and help to retain in the furnace that part of the finer Mesabi ore otherwise likely to be blown over. It is difficult to show in concrete figures just how much loss is occasioned directly by waste of part of the metallic contents of the charge as flue-dust, and indirectly by the more rapid deterioration of furnace- and stove-linings, together with the added

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cost for repairs to gas- and blast-mains and castings. Possibly too much attention has been paid to the analysis of ore charged into the furnace, and too little attention paid to the actual furnace-yield.

The South Range ores are well adapted chemically to the manufacture of foundry, gray-forge, and basic pig-iron. The impurities common to all iron-ores, namely, silica, alumina, phosphorus, and manganese, are reasonably uniform and not excessive.

We think that furnace-men generally will agree that iron-ores containing limonite and iron carbonate, as these ores do, will absorb carbon more readily in the blast-furnace, due to the expulsion of water and carbonic acid in the heating- and reducing-zones, thereby opening additional channels for the reducing gases and effecting reduction throughout the charge. This feature should promote fuel-economy and faster driving of the blast-furnace.

Aside from a slightly higher silica-content than is really necessary, and a consequently larger slag-volume, there is really no objectionable feature to these ores. The problems of the future in the metallurgy of iron certainly include the use of leaner iron-ores. Is it not better gradually to widen the supply by including now in the furnace-mixture such ores as are herein described, than to defer their use until necessity compels, as it surely must, a mixture of all low-grade ore?

Is it not possible, by means of larger furnaces and more economical machinery for handling materials into and out of the furnace, so to reduce costs as to overcome the added charge for flux, and possibly fuel, necessitated by use of leaner ore?

These South Range ores are available, their recovery offers no great difficulties, the deposits necessitate no great outlay to open up, the freight-rate to Duluth and Superior is the same as the Mesabi rate, and competent mining-men estimate the cost to place this ore on board cars, exclusive of royalty, at $1.25 per ton. Nowhere else in the Lake Superior region is there so great a tonnage of merchantable iron-ore awaiting development.

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A Titaniferous Iron-Ore Deposit in Boulder County, Colo.

BY E. P. JENNINGS, SALT LAKE CITY, UTAH.

(Cleveland Meeting, October, 1912.)

LARGE deposits of titaniferous iron-ore occur at Caribou, an old silver-mining camp in Boulder county, Colo., 17 miles west by south of Boulder, and a few miles northwest of the tungsten-mines. Professor Regis Chauvenet[1] published a part analysis of the ore many years ago, but gave no description of the wall-rock.

Caribou hill, which contains both the silver-bearing veins and the iron-ore deposits, is a low dome of gabbro surrounded by the Archæan schists and gneisses of the Front range of the Rocky mountains, the peaks of which, a few miles west of Caribou, attain an elevation exceeding 12,000 ft. The gabbro stock is roughly elliptical in outline, with approximate diameters of 1 and 3 miles. The crest of the dome is 10,500 ft. above sea-level, and the town and ore-deposits are 500 ft. lower. Outcrops of the iron-ore occur in the town and extend NW. for 2,000 ft., where they disappear under the swampy meadows of Caribou park. Portions of the deposit are covered by drift, but one continuous outcrop is 900 ft. long and from 200 to 250 ft. wide. A second deposit is 0.25 mile south of the town along the eastern slope of the hill. It is not as well defined as the north bed, but is approximately 1,000 ft. long and from 100 to 200 ft. wide. The easily disintegrated ore does not form prominent outcrops, but the outlines of the deposits are readily traced by the large amount of ore fragments that cover the ground. Trenches and pits sunk on the outcrop expose a roughly banded magnetic iron-ore, with pyroxene as the principal gangue-mineral.

The Gabbro.

This rock was classified in the field as a gabbro from the presence of pyroxene and striated feldspars, and for convenience the field-name will be retained.

[1] Chauvenet, Regis, *Biennial Report of State School of Mines*, Golden, Colo., p. 16 (1886).

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Macroscopic Characters.

The rock is light to dark gray, even-grained and granular. It weathers light brown. It is medium-coarse in texture in the central portions of the stock, and fine-grained near the contacts with the gneiss, but there is no sharp line of demarcation between the two varieties. The silver-veins occur in the coarse-grained variety, and the iron-ores in the finer-textured rock.

Under a lens both varieties show abundant striated feldspars, white orthoclase, dark-green pyroxene, iron-ores, and brown biotite. The latter mineral is abundant in the coarse-grained rock, but scanty or entirely absent in the fine-grained phase.

Microscopic Characters.

Thin sections (Figs. 1 and 2) show the following minerals: Iron-ores, apatite, pyroxene, biotite, plagioclase, and orthoclase. The plagioclase is in clear lath-shaped crystals that in the fine-grained rock average from 1 to 2 mm. in length, twinning after the albite and Carlsbad laws and containing scanty apatite inclusions. In the coarse-grained variety, the feldspars are from 3 to 5 mm. long, and the other minerals proportionately larger. Determinations by the statistical method and extinction angles in zone at right angles to 010, where albite and Carlsbad twinning are combined, give: Ab 1.5, An 1 to Ab 1, An 1. These ratios make the plagioclase a variety in the andesine series on the border with labradorite. Orthoclase, which is the last mineral to separate, occurs in irregular forms, filling spaces between the other minerals; it is clear to milky and contains abundant inclusions of ore and pyroxene. Colorless to pale yellowish green, non-pleochroic grains and irregular masses of pyroxene form about 10 per cent. of the rock. Maximum extinction angles of 39° to 40° from C. axis place the mineral near diopside. Biotite in shreds and plates, closely associated with the diopside, forms about 3 per cent. of the coarse-grained rock, but is often absent in the fine-grained variety. Stout prisms and hexagonal grains of apatite are rather abundant as inclusions in all the other minerals. Iron-ore is abundant as inclusions in the diopside and orthoclase, but is absent from the plagioclase.

The order of crystallization is: apatite, plagioclase, ores, biotite, diopside, orthoclase.

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The analyses given in Table I. were made by the methods recommended by Hillebrand[1] of fresh material taken from shafts and tunnels:

TABLE I.—*Analyses of the Gabbro.*

	I. Coarse-Grained Rock from Shafts Near Center of the Gabbro Stock, Contains Small Amount of Biotite.	II. Fine-Grained Rock from Tunnel Near Ore-Bed, Northern Edge of Gabbro.
	Per Cent.	Per Cent.
SiO_2	51.43	52.10
Al_2O_3	19.35	19.35
Fe_2O_3	5.32	4.63
FeO	4.70	4.16
MgO	3.05	3.19
CaO	7.09	7.15
Na_2O	3.54	3.39
K_2O	3.52	3.77
H_2O + 105°	0.35	0.13
H_2O — 105°	0.08	0.06
TiO_2	0.85	0.94
P_2O_5	0.60	0.71
MnO	Tr.	0.10
	99.88	99.68

The analyses prove the rock to be a gabbro closely related to the syenites. The high potash percentage with the consequent orthoclase and biotite remind one of the gabbros at Duluth, Minn., and others in the Cortlandt series near Peekskill, N. Y. When recast according to the methods of the quantitative system for the classification of igneous rocks, the following norms or theoretical percentages of different components are obtained:

I.			II.	
Orthoclase	20.57		22.24	
Albite	29.34	76.60	28.82	77.19
Anorthite	26.69		26.13	
Diopside	4.23		4.20	
Hypersthene	6.32		7.02	
Olivine	1.56		0.90	
Magnetite	7.66	22.63	6.73	22.20
Ilmenite	1.52		1.67	
Apatite	1.34		1.68	
H_2O	0.43		0.19	
	99.66		99.58	

[1] Hillebrand, W. F., The Analysis of Silicate and Carbonate Rocks, *Bulletin No.* 305, *U. S. Geological Survey* (1907).

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FIG. 1.—FINE-GRAINED GABBRO, 24 DIAMETERS; POLARIZED LIGHT; NICOLS CROSSED. ACTUAL FIELD, 3 MM. PLAGIOCLASE (PG), ORTHOCLASE (OR), PYROXENE (PY), MAGNETITE (M).

FIG. 2.—COARSE-GRAINED GABBRO, 18 DIAMETERS; POLARIZED LIGHT; NICOLS CROSSED. ACTUAL FIELD, 4 MM.

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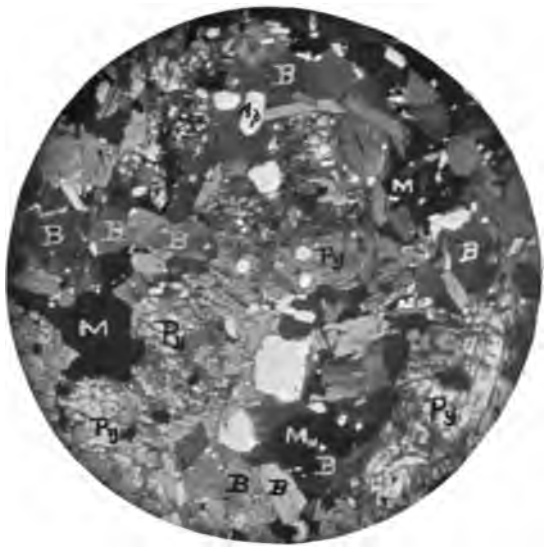

FIG. 3.—BIOTITE-PYROXENE ZONE OF DIKE, 9 DIAMETERS; ORDINARY LIGHT. ACTUAL FIELD, 8 MM. APATITE (AP), BIOTITE (B), PYROXENE (PY), MAGNETITE (M).

FIG. 4.—OUTER EDGE OF DIKE, 9 DIAMETERS; ORDINARY LIGHT. ACTUAL FIELD, 8 MM. PLAGIOCLASE (PG) IS LAST MINERAL TO SEPARATE.

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FIG. 5.—ORE OF AVERAGE COMPOSITION, 9 DIAMETERS; ORDINARY LIGHT. ACTUAL FIELD, 8 MM. THE LIGHT-COLORED AREAS ARE PYROXENE.

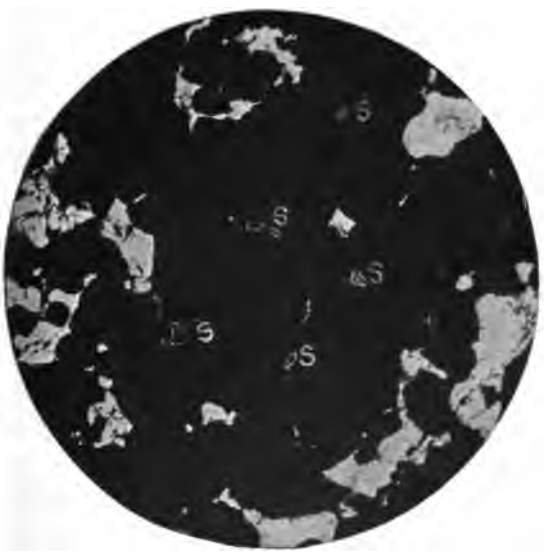

FIG. 6.—ORE FROM CENTRAL ZONE OF DIKE, 18 DIAMETERS; ORDINARY LIGHT. MAGNETITE, DIOPSIDE, SPINEL (S).

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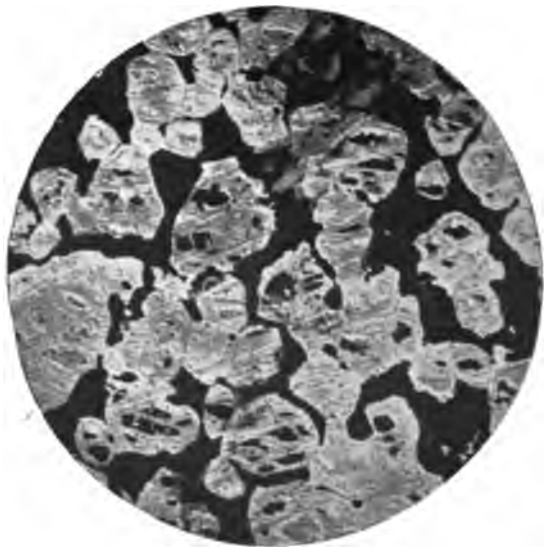

FIG. 7.—ORE FROM SURFACE, DIOPSIDE ALTERED TO SERPENTINE, 24 DIAMETERS; POLARIZED LIGHT; CROSSED NICOLS. ACTUAL FIELD, 3 MM.

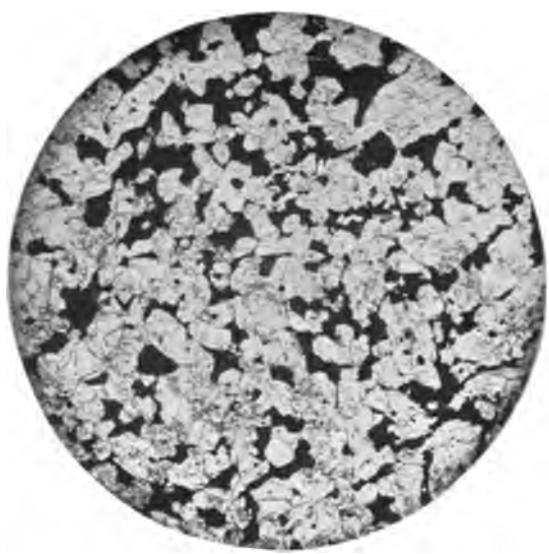

FIG. 8.—LEAN ORE, PYROXENE ZONE OF DIKE, 9 DIAMETERS; ORDINARY LIGHT. ACTUAL FIELD, 8 MM.

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	I.	II.
$\frac{Sal}{Fem}$	$= \frac{76.60}{22.63} = 3.38 =$ Class II.	$\frac{77.19}{22.2} = 3.47 =$ Class II.
$\frac{F}{Q}$	$= \frac{76.60}{0} => \frac{7}{1} =$ Order 5	$\frac{77.19}{0} => \frac{7}{1} =$ Order 5
$\frac{Na_2O' K_2O}{CaO'}$	$= \frac{93}{96} = 0.96 =$ Rang 3	$\frac{95}{94} = \frac{1.01}{} =$ Rang 3
$\frac{K_2O'}{Na_2O'}$	$= \frac{37}{56} = 0.66 =$ Subrang 3	$\frac{40}{55} = 0.72 =$ Subrang 3

The magmatic symbol for both rocks is: II. 5. 3.3, or Shoshonose, by the chemical classification. I would classify the rock as an orthoclase-biotite gabbro in preference to calling it a monzonite, on account of the predominance of the plagioclase.

The Iron-Ores.

The principal ore-deposit is a pyroxene-magnetite-biotite dike, 250 ft. wide, in which the minerals are very unequally distributed, the magnetite being largely concentrated in the central portions, while the biotite is confined to a zone near the walls.

The order of distribution from the walls to the center is: 25 ft. of biotite-pyroxene rock (Figs. 3 and 4), with accessory plagioclase, magnetite, and orthoclase; 50 ft. of pyroxene rock, with little magnetite and biotite and no feldspar; the remaining 50 ft. to the center of the dike is a low-grade iron-ore, averaging from 30 to 35 per cent. of iron, and containing about equal amounts of magnetite and pyroxene. Samples can be selected from this central zone that are nearly pure magnetite, but the rich bands are but a few inches wide and are difficult to separate from the leaner ore. There is no sharp division between the different zones, but a gradual merging of one into the other.

The ore, which is strongly magnetic, is an intimate mixture of titaniferous magnetite and pyroxene, with accessory spinel, apatite, and biotite. Its texture varies from a coarse-grained granular aggregate to one extremely fine-grained, that has a slaty structure resembling the slate ores of Lake Superior.

Thin sections (Figs. 5, 6, 7, and 8) show abundant stout prisms and grains of colorless to slightly yellowish green pyroxene, from 0.1 to 5 mm. in length, often in well-defined crystals, distributed quite uniformly in the magnetite ground-mass.

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They contain a few inclusions of biotite and magnetite (or ilmenite) and needles of apatite; are non-pleochroic, and have a maximum extinction angle of 40°, which places the mineral near diopside. A part alteration of the diopside to serpentine is common in the surface-ore.

Dark-green isotropic crystals of quadratic outline and high refractive index, seldom over 0.1 mm. in diameter, occurring as rather abundant inclusions in the magnetite, were identified as spinel. In 24 sections examined, 12 contained spinel. It appears to be confined to the richer ores, and is always absent from those sections in which the diopside crystallized ahead of the magnetite. Apatite in stout prisms and hexagonal grains occurs as inclusions in both the magnetite and pyroxene, but is relatively more abundant in the lean ore in which the pyroxene predominates. The order of crystallization in the lean ore is: apatite, biotite, diopside, magnetite. In ore containing about 70 per cent. of magnetite, the diopside and magnetite crystallized simultaneously, but apatite and spinel were the first to separate. In the rich ore the order of crystallization is: apatite, spinel, magnetite, diopside. The composition of the eutectic is probably near 70 per cent. of magnetite (including spinel) and 30 per cent. of diopside.

Attempts were made to separate ilmenite from magnetite by means of an electro-magnet without success, as were also experiments in etching polished surfaces with dilute HCl.

The inference that the ilmenite is very finely divided and intimately mixed with the magnetite, is strengthened by the easy and complete solubility of the iron oxides in hot dilute HCl, that is in marked contrast to the difficultly soluble ores of Minnesota and Canada, in which the ilmenite occurs in grains of appreciable size. The ore is strongly magnetic, and many of the surface-boulders are natural load-stones.

Analyses of the Ore.

The following method was used in analyzing the ore: 5 g. of ore was treated with hot HCl, which dissolved all the magnetite and ilmenite; the solution was filtered from the insoluble residue; after which the filtrate was analyzed by the methods of the chemists of the U. S. Steel Corporation. The insoluble residue, consisting of diopside, spinel, and small amounts of TiO_2, was fused with $Na_2S_2O_7$, so as to decompose

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the spinel and TiO_2, but not the diopside; the melt was dissolved in hot dilute HCl and filtered from the insoluble diopside; the TiO_2 was separated and was added to the TiO_2 obtained in the portion soluble in HCl. The SiO_2, Al_2O_3, Fe_2O_3, and MgO were determined, and the results obtained were taken as the composition of the spinel. The diopside was analyzed by the ordinary methods for silicate minerals. Direct determinations of FeO were made in the portion soluble in HCl by titration with bichromate.

There may be some question as to the accuracy of the method of separating the minerals, but the presence of only sufficient lime to combine with P_2O_5, and its entire absence in the spinel-analyses, is evidence that the diopside is not attacked, either by the HCl or $Na_2S_2O_7$.

The Al_2O_3, SiO_2, and MgO found in the portion soluble in HCl are believed to be parts of the magnetite molecule, and not derived from either the spinel or the diopside.

Table II. gives the analysis of medium-grade ore from the north deposit.

TABLE II.—*Composition of Medium-Grade Ore From North Deposit.*

		Per Cent.		
Soluble in HCl	FeO with TiO_2	4.02	Ilmenite,	8.50
	TiO_2	4.48		
	FeO	20.10	Magnetite,	67.25
	Fe_2O_3	44.72		
	SiO_2	0.31		
	Al_2O_3	0.81		
	MgO	1.31		
	CaO	0.10	Apatite,	0.18
	P_2O_5	0.08		
Portion decomposed by $Na_2S_2O_7$	SiO_2	0.20	Spinel,	6.70
	Al_2O_3	4.29		
	FeO	0.53		
	MgO	1.68		
By fusion with Na_2CO_3	SiO_2	9.09	Diopside,	16.89
	Al_2O_3	1.17		
	FeO	0.66		
	CaO	3.65		
	MgO	2.17		
	Na_2O	0.15		
	K_2O	Tr.		
	$H_2O + 105°$	0.51		0.51
		100.03		100.03

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	I. Spinel, Calculated from Ore-Analysis.	II. Diopside, Calculated from Ore-Analysis.	III. Spinel, Direct Analysis of Separated Material.	IV. Diopside, Direct Analysis of Separated Material.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
SiO_2	2.98	53.82	2.96	54.24
Al_2O_3	64.03	6.93	64.28	7.03
FeO	7.91	3.90	7.67	3.97
CaO	.....	21.62	.....	21.41
MgO	25.08	12.85	24.46	12.65
Na_2O	.....	0.88	.....	0.87
K_2O	.....	T	.....	Tr.
	100.00	100.00	99.37	100.17

The spinel is the iron-magnesian variety, pleonaste. The diopside is high in Al_2O_3, but its extinction angle of 40° is that of diopside. Many part analyses were made to determine the ratio of TiO_2 to Fe, resulting in an average of 1 to 10.7, the highest being 1 to 11.4, and the lowest 1 to 8.5. Professor Chauvenet's analysis differs greatly from these results, being TiO_2, 36.76; and Fe, 36.93 per cent.

These interesting ore-deposits have little or no economic importance, but are excellent examples of iron-ores of igneous origin. They may be considered either as true dikes, or as segregations in place from the wall-rocks. The first supposition is the most probable, for the following reasons: In form the deposits are typical dikes of great length and uniform strike, in marked contrast with the irregular outline of most segregated ore-bodies; the wall-rock contacts are sharp and well defined; there is no local impoverishment of the gabbro in ferro-magnesian minerals near the dike, the ratio of these to the feldspars being essentially the same in rock near the dike as in rocks from points 1,000 ft. away, indicating no concentration of basic minerals from the wall-rock.

Similar occurrences have been described as dikes by Kemp,[3] in New York State, and Ball,[4] in Wyoming. Also, Vogt[5] classes certain Norwegian deposits as true dikes.

[3] Kemp, J. F., Titaniferous Iron Ores of the Adirondacks, *Nineteenth Annual Report, U. S. Geological Survey*, part iii., p. 414 (1897–98).

[4] Ball, Sydney H., *Bulletin No.* 315, *U. S. Geological Survey*, p. 206 (1907).

[5] Vogt, J. H. L., *Om dannelse af de vigtigste i Norge og Sverige representerede grupper af jern-Malmforekomster*, p. 14 (Christiania, 1892).

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The inference is that the origin of the dike was in a deep-seated segregation of ferro-magnesian minerals and magnetite, due to a differentiation from the gabbro magma, and that its composition is essentially the same as that of the parent basin.

The later rearrangement of the minerals of the dike, due to slow cooling under a great mass of superimposed rock, has partly concentrated the magnetite near the center and the biotite along the walls.

The order of the deposition of the minerals corresponds closely with the order of crystallization observed in thin sections of the ore, the biotite being among the first minerals to separate, and magnetite with the last in each case.

The original dike-magma contained an excess of Al_2O_3 and MgO above that required to form pyroxene and biotite; this excess separated as spinel, excepting small amounts that passed into the magnetite molecule.

Olivine, a common constituent of gabbro and titaniferous ores, is notably absent, though not from any lack of MgO in the magma.

Clinton Iron-Ore Deposits in Kentucky and Tennessee.

BY S. WHINERY, NEW YORK, N. Y.

(Cleveland Meeting, October, 1912.)

I AM indebted to L. E. Bryant, of Danville, Ky., President of the Virginia Mining Co., operating coal-mines in Scott county, Tenn., for the following information relating to the existence of the Clinton iron-ore under the NW. slope of the Cumberland Plateau in Kentucky and Tennessee.

During the past 15 years quite an important oil-field has been developed in a region having Wayne county, Ky., as its center, and a large number of wells have been drilled in that and adjacent territory in the search for oil. Many of these wells have been carried to considerable depths below the recognized oil-bearing horizon, and one of them was drilled by Mr. Bryant at Pine Knot, a station on the Cincinnati, New Orleans & Texas Pacific railroad, a few miles north of the point where that road crosses the Kentucky-Tennessee State-line. The elevation of

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the surface at the crossing is about 1,400 ft. above sea-level. The railroad is located on the dividing ridge between the Cumberland river and Big South Fork. At a depth of 1,720 ft. below the surface the drill reached and passed through a formation which the drillers called "red rock," but which Mr. Bryant recognized as Clinton iron-ore, as its geological position corresponded to that formation. The ore at this point was 6 ft. thick and appeared to be of uniform quality throughout. An analysis of the drillings gave: Fe, 33; Mn, 0.25; SiO_2, 3; Al_2O_3, 2; CaO, 13; MgO, 7; P_2O_5, 0.333 per cent.; S, trace.

This analysis seems to confirm the belief that the ore is Clinton, and that it is of a quality equal to the average of the deep or unbleached Clinton ores in Tennessee and Alabama.

This discovery led Mr. Bryant to make further examinations and inquiries, and he found that the stratum called "red rock" by the drillers had been passed through and noted in the great majority of the wells carried down to the same geological horizon, over quite a wide area in and around the oil-region. Its geological position was readily fixed in these wells, as it was noted as being from 50 to 100 ft. below the Devonian black shale, which is very persistent in the region, is universally recognized and referred to by the well-drillers. The thickness of the "red rock" was not usually recorded, but was stated by the drillers to be from 3 to 6 ft.

In a number of wells reaching the horizon of the ore its occurrence was not noted. This is not surprising in view of the well-known lenticular formation of the Clinton ore-deposits in the South.

These facts seem to indicate that rich and comparatively thick deposits of Clinton ore underlie the Cumberland Plateau in this region and extend under its northern slope. If so, it will, undoubtedly, in time, become commercially very valuable, particularly as the whole region is also underlain by three or four workable beds of coal, at least two of which are good coking-coal, so that both ore and coal could be mined from the same or adjacent shafts. The coal outcrops above the surface of the larger streams and is being mined in large quantities by drifts in the region. Shafts sunk along the streams in the vicinity would reach the iron-ore at a depth of about 1,000 feet.

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The Ta-yeh Iron-Ore Deposits, Hu-pei Province, China.

BY C. M. WELD, LOW MOOR, VA.

(Cleveland Meeting, October, 1912.)

In the course of my professional work in China during the fall of 1907, I had an opportunity to visit the iron-ore mines at Ta-yeh in Hu-pei province (long. 114° 50′ E., lat. 30° 20′ N.). It occurs to me that a rescript and discussion of my notes taken at that time should be of interest, particularly as supplementing the exceedingly interesting recent paper of Thomas T. Read on The Mineral Production and Resources of China.[1]

Ta-yeh is situated near the right bank of the Yangtze river. The mines are at an average distance of about 16 miles from the river-port Shi-hui-yao, which in turn is some 70 miles SE. of Hankow. The ores are shipped to the works of the Han-Yeh-P'ing Iron & Coal Co. at Hanyang, as also very largely to the Japanese government iron- and steel-works at Wakamatsu, Japan. Smaller quantities have been exported to the Pacific coast of the United States.

These ore-deposits have been briefly noticed in a number of publications, but, so far as I know, Mr. Read's description, in his chapter on the iron-ores of the Chinese Empire,[2] contains the first attempt to define their geological character. In his more recent paper Mr. Read follows very closely the text of his earlier description. To quote briefly from the recent paper, he says: "The iron-ores at Ta-yeh . . . . . lie along the contact between a marble and an intrusive body of a dark-gray syenitic rock." He refers to a figure which exhibits in plan, but not in section, the relative positions of the syenite, ore-bodies, and limestone. The contact is seen to trend approximately E–W., with the syenite on the north and the limestone on the south. "The syenite lies to the north of the marble, and a few miles farther north the marble again appears, iron-ore again being present along the contact." No litho-

[1] *Trans.*, xliii., 3 to 53 (1912).

[2] *The Iron Ore Resources of the World*, vol. ii., p. 913 (Stockholm, 1910).

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logical description of the syenite is given. The limestone is discussed as probably of Carboniferous age, and the suggestion is advanced that "this limestone is a horizon for iron-ores in the Yangtze valley, as they are associated with it in notable amount near Nanking." Again: "In Kiang-su province iron-ores are widely distributed, occurring chiefly in the region about Nanking in association with the limestone previously mentioned." And in the following paragraph: "In the province of An-hwei the geological conditions are similar to those in Kiang-su and Hu-pei." In describing the ore he says: "The ore is a good quality of hematite [here follow analyses] . . . . At one place it is slightly magnetic, apparently having been partly reduced to the magnetic oxide by the action of reducing solutions, which have deposited small amounts of copper- and iron-sulphides along the foot-wall."

A. J. Seltzer[3] has described the Ta-yeh deposit as "a contact deposit, the contact on the north wall being a compact, close-grained diorite, the south contact being a pure, almost white, marbly limestone." He presents analyses for three kinds of ore, namely, hematite, magnetite, and manganiferous brown ore.

Another reference which I have before me[4] describes the Ta-yeh ores as including magnetite, hematite, and manganiferous brown ores, in a steeply-dipping vein. "On the section of the deposit where lower phosphorus ores are found, the vein is in contact with almost pure limestone in very large quantities."

My visit to the Ta-yeh mines was only of a few hours' duration; my examination was therefore necessarily superficial, and confined to a single one of the several ore-bodies which have been developed. I gathered no samples or specimens, and have therefore to rely wholly upon macroscopic inspection, made at the time, of the ores and associated rocks and minerals, having neither analyses nor thin sections for further detailed study.

Turning to my notes, I find the ore-bodies described as occurring along an approximately E–W. contact between horn-

[3] Iron and Steel Works at Hanyang, Hupe, China, *Engineering and Mining Journal*, vol. lxxxix., No. 24, p. 1232 (June 11, 1910).

[4] The Iron Industry of China, *Iron Age*, vol. lxxxi., No. 19, p. 1436 (May 7, 1908).

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blende-granite and limestone, the former lying north and the latter south of the contact.

The granite is a granitoid rock consisting essentially of quartz, potash-feldspar, and hornblende. While ordinarily holocrystalline to the naked eye, some fine-grained and porphyritic facies appear at the contact. The general color-effect on a slightly weathered surface is light gray, peppered with black.

The limestone is a light-gray smooth-textured rock, changing to a coarsely crystalline and often pure white marble at the contact. Seltzer[5] states that this stone carries 96 per cent. of $CaCO_3$. At least it appears to be sufficiently pure to yield an acceptable flux for the Hanyang blast-furnaces, and is quarried for that purpose at a point about 600 yd. south of the contact. Contact-minerals in the limestone appear to be rare; in fact, none were observed except where occasional areas were seen to be abundantly impregnated with minute resinous-brown garnets, frequently arranged in bands. It should be noted, however, that my remarks apply to a section where the granite and the limestone are separated by about 200 ft. of iron-ore; no actual contact between the two came under my observation.

In structure the limestone exhibits a great anticline in the country south of the contact, its axis trending approximately ENE–WSW. The northward-dipping leg of this anticline is bent sharply back upon itself as it approaches the contact. The granite occupies, so far as I could learn, an elliptical area, from 6 to 10 miles E–W. by from 2 to 3 miles N–S. Within this area the granite peaks stand out in somewhat bolder relief than the surrounding limestone country. I have already quoted Read to the effect that the limestone reappears to the north. On the east limestone is seen along the right bank of the Yangtze river, except where overlain by red beds.

Before turning to the ore-bodies it will be convenient to refer briefly to my figures, which have been reproduced from the original sketches included in my notes. Fig. 1 represents the district as a whole, and shows the general position of the mines with reference to the Yangtze river. Fig. 2 represents the western portion of Fig. 1 on an enlarged scale, while Fig. 3 is a geological cross-section taken on a N–S. line and looking

[5] *Op. cit.*

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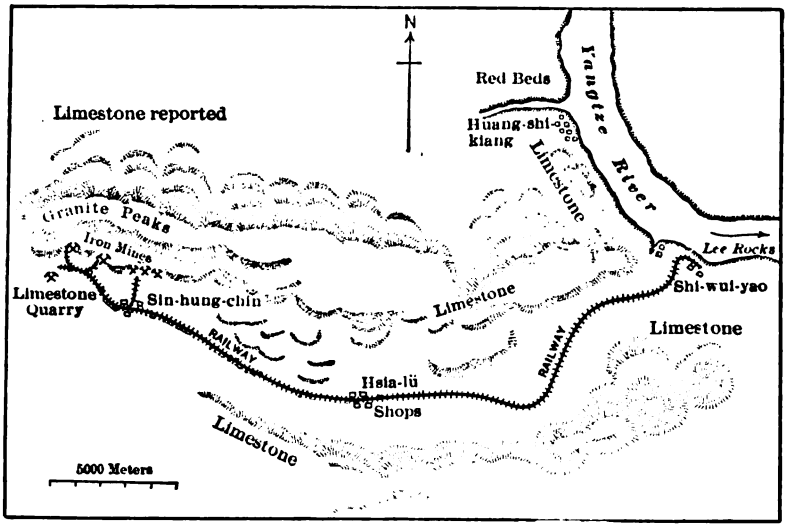

FIG. 1.—SKETCH-MAP SHOWING POSITION OF THE TA-YEH IRON-MINES, HU-PEI PROVINCE, CHINA.

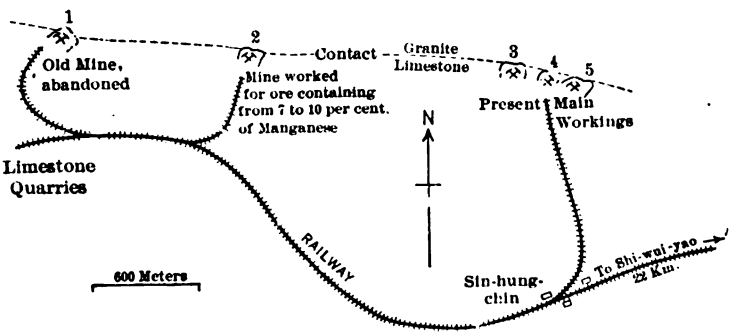

FIG. 2.—SKETCH-MAP OF THE TA-YEH IRON-MINES, HU-PEI PROVINCE, CHINA.

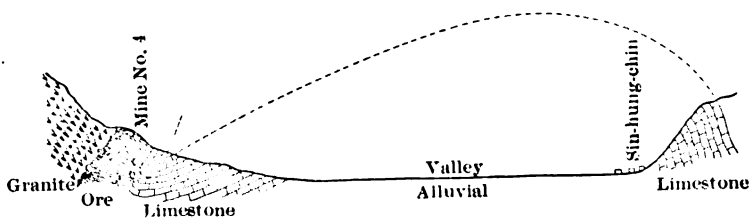

FIG. 3.—GEOLOGICAL CROSS-SECTION THROUGH MINE NO. 4 AND SIN-HUNG-CHIN, LOOKING EAST.

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east, through mine No. 4 and the hamlet Sin-hung-chin. All the above sketches are only approximately to scale.

These illustrations show five mines, numbered consecutively from west to east. Up to the time of my visit there had been no deep mining, all workings being comparatively shallow open-cuts above drainage. Mine No. 1 proved to be on a small deposit, and had been worked out and abandoned. Mine No. 2 was still worked, but in a small way only, the ores being largely manganiferous brown ores. Mines Nos. 3, 4, and 5 were closely grouped together, apparently on a common lens of ore, and constituted the main workings in 1907. Explorations further to the east had discovered additional ore-deposits, but these had at that time not been developed. Read, as quoted above, says that ores have also been found along the northern contact of the granite and limestone.

We note here that the contact is only intermittently ore-bearing, the deposits occurring in a succession of lenses of varying size, with barren stretches between.

I examined briefly the ore-body revealed in mines Nos. 3, 4, and 5. This lens was from 1,500 to 2,000 ft. long and from 100 to 200 ft. thick, and had been excavated to a maximum depth of probably not more than 75 ft. below the original crop. The strike is approximately E–W., with the contact, and the dip from 65° to 85° N. Owing to the northward dip, the granite, in mining terminology, constitutes the hanging-wall, and the marble is the foot-wall. The granite wall, while fairly persistent in its E–W. strike in a general way, is irregular and somewhat angular in detail. The actual line of contact between the ore and the granite, where it could be observed in a fresh condition, was seen to be sharp. On the foot-wall side the transition-line from ore to marble was again sharp, and even more irregular in detail than on the hanging-wall. In this case the irregularities were curved rather than angular, however, and close to the wall many rounded inclusions of marble were seen in the ore.

The ores filling the space between the two walls were for the most part massive, though a certain amount of shearing parallel with the contact was observed. They include hematite and magnetite, the latter being present in a very considerable proportion, and being, in my opinion, primary. Further,

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there are smaller amounts of the hydrated oxides in the more weathered portions of the vein. The mass, as a whole, is high grade, gangue-minerals appearing in rather subordinate amounts. These include quartz, some amphibole and greenish mica, occasional epidote, and more commonly copper carbonates, with rare sulphides of copper and iron. In Table I. I have brought together a number of analyses from various sources.

TABLE I.—*Analyses.*

	(1)	(2)	(3)			(4)	(5)
			a.	*b.*	*c.*		
Fe	58 to 68	60 to 62	62.0	48.0	65.0	59 to 65	65.5
SiO_2	7 to 3	5 to 3	4.0	6.0	3.0	10 to 3.5	3.5
Al_2O_3	1 to 2	1 to 2	2.0				
Mn	0.2 to 0.4	0.2 to 0.4	0.3	7.0	0.13	.....	0.10
P	0.04 to 0.25	0.05 to 0.25	0.05	0.03	0.12	0.04 to 0.10	0.04
S	0.05 to 0.10	0.05 to 0.12	0.02	0.01	0.06	0.10	0.05
Cu	0.05 to 0.25	0.05 to 0.25	.....	.....	.....	0.20	0.09

(1) From *Iron Age*, (4) p. 1436. (2) Read, (1) p. 320. (3) Seltzer, (3) p. 1233.—*a*, hematite; *b*, brown ore; *c*, magnetite. (4) Reported to me by officials at the Hanyang iron- and steel-works. (5) Reported to me by officials at the Japanese government iron- and steel-works.

The evidence at hand bearing upon the geological horizon at which the Ta-yeh deposits occur consists briefly of the following facts:

The limestone associated with the ore-deposits is closely overlain by red beds. These appear at Huang-shi-kiang, Fig. 1, and continue thence for some distance up the river. There are also conspicuous red cliffs several miles down the river. Furthermore, thin coal-beds are reported to exist within a few miles NE., W., and SE. of Ta-yeh.

Willis[6] gives the following geological column for the Yangtze gorges, about 200 miles westward:

Permo-Mesozoic: K'ui-choú series.

1,000 ft. sandstones, shales, and sandy shales with thin limestones and coal-seams.

150 ft. gray fossiliferous limestone.

400 ft. massive red shale.

Unconformity inferred.

[6] *Research in China*, Willis, Blackwelder, and Sargent, vol. i., part i, p. 264 (Carnegie Institution, 1907).

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Upper Carboniferous: Wu-chan limestone.

3,000 ft. massive thick-bedded limestone of gray-brown or black color; locally flinty.

1,000 ft. shales and limestones, the latter locally rich in flint nodules.

Below this come 1,800 ft. of Sin-t'an shale (Middle Palæozoic), and about 4,000 ft. of Sinian (Cambro-Ordovician) Ki-sin-ling limestone, with some minor beds of shale and conglomerate, overlying the basal Algonkian (?) gneisses.

Elsewhere Willis cites Kingsmill with reference to localities near the Yangtze river as follows:[7] ". . . he [Kingsmill] distinguishes two horizons at which coal beds occur; one, the 'lower coal and iron shales,' 600 to 800 feet thick, being near the middle of his 'Tung-t'ing' series and dividing the great Carboniferous limestone in two; the other, the upper member of his 'Chung-shan' series, being higher up in the geologic column and in all probability of the same age as the early Jurassic (Rhetic) coals of the Middle Yang-tzi."

Comparing the above sections with the observed and reported conditions in the vicinity of Ta-yeh, it seems a reasonable inference that the Ta-yeh limestone is near the top of the Wu-chan (Upper Carboniferous); that the red beds at Huang-shi-kiang as well as down the river correspond to the "massive red shale" at the base of the Permo-Mesozoic; and that the adjacent thin-coal areas, which practically surround the granite ellipse, belong to the Permo-Mesozoic (early Jurassic) coals of Willis, corresponding to the "Chung-shan" coals of Kingsmill. This inference would argue a quaquaversal dip of the sediments about the granite.

In connection with this inquiry into the probable age of the associated limestone, I find of interest a description of certain Kiang-su province iron-ore deposits, written 50 years ago by Markham, then British consul at Chin-Kiang, a town on the Yangtze lying about 35 miles ENE. from Nanking. This description has been preserved by Kingsmill and is quoted by Willis.[8]

Markham describes three localities where iron-ore has been

[7] Mineral Resources of China, *Economic Geology*, vol. iii., No. 1, p. 32 (Jan.–Feb., 1908).

[8] *Loc. cit.*, p. 35.

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found. One of these, an inferior deposit within 3 miles of the gates of Nanking, is definitely referred to beds "at the foot of the Chung-shan." The other two, being respectively 5 and 14 miles distant from Chin-Kiang, appear to be probably below Kingsmill's "lower coal and iron-shales." While, therefore, the first is at about the same horizon as the Ta-yeh ore-bodies, the others may be from 1,000 to 1,500 ft. lower down in the geologic scale, though still in the Upper Carboniferous. Both these last are described in association with chert-bearing limestone, in one case there being present a large intrusive mass of porphyry.

My brief inspection of the Ta-yeh deposits was quite insufficient to warrant any attempt on my part at an estimate of the probable available tonnage of ore, though this would appear to be very considerable. Mr. Read and others have presented descriptions and photographs of the generally primitive methods of open-cut hand-mining, and I find little in my notes to add to what has already been written in this direction. I, therefore, turn at once to the question of genesis.

In quoting from Read I have tried to include all his more significant references to the ore-deposits from a possible genetic standpoint. Whereas, the contact is recognized as being between an intrusive igneous rock and an intruded sediment, the genesis of the ores would appear to be referred rather to some cause other than the intrusive. There is no definite statement under this head, but two things are suggestive of such a trend of view, namely: the point is brought out that the deposits of Hu-pei (Ta-yeh), An-hwei, and Kiang-su provinces are probably at a common stratigraphical horizon; the magnetite is referred to as secondary. The first of these would naturally suggest that there is held to be some definite horizon in the limestone which, due to peculiar inherent or closely associated conditions, would carry ore-bodies to the exclusion of other horizons; in other words, that the ore is due to the presence of a definite horizon in the limestone, and not to the accidental association of this or any other horizon of limestone with an intrusive. The second point would suggest the opinion that the ore has been derived from meteoric, rather than deep-seated, sources.

There can be little doubt as to the nature of the contact. It is decidedly not a natural one. If faulted, then a fault with

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a throw of several thousand feet would be required to bring the Upper Carboniferous limestone into juxtaposition with the basement Archæan. So profound a fault would surely have left a corresponding stamp upon the adjacent strata. Such shearing as was observed was in the ore-body and, therefore, due to movement subsequent to its deposition. The granite gives no evidence of profound faulting, and the overturning of the limestone strata at the contact can be more readily accounted for otherwise. The contact is, in fact, undoubtedly one of intrusion, and in the granite we find a great laccolith which has invaded the limestone at some depth, and has since been uncovered by erosion. The elliptical area of the granite and the apparent quaquaversal dip of the sediments are in line with this supposition, while the structural conditions actually observed along the southern contact can well be explained by it. The limestone shows evidence of heat-metamorphism. The rarity of contact-minerals in the zone which came under my observation may be considered strange, but, as already pointed out, the intrusive and sedimentary members were separated at that point by 200 ft. of iron-ore. A study of the contact along the stretches barren of ore should be full of interest.

The source of the iron which has been deposited along this intrusive contact is our next consideration.

That the iron has been derived from the limestone, or from any particular horizon in the limestone, or that the ore-bodies represent the metamorphosed upturned edge of bedded veins of ore original in and conformably intercalated with the limestone, are suppositions which appear to me highly improbable. There is no appearance of such a possible source for the iron, or original existence of the vein, either in the character of the limestone beds, or in their position relative to the deposits. Read's suggestion that the deposits of the three provinces, Hupei, An-hwei, and Kiang-su, are at a common stratigraphical horizon would, if true, lend support to some such view. I have indicated, however, that the ores of the several provinces are more probably not at a common horizon.

A more plausible hypothesis, in my opinion, would be that the iron of Ta-yeh has been derived out of the once overlying but now eroded red beds, and has been transported in solution

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by descending meteoric waters along trunk-channels developed at the contact, till deposited as opportunity favored, either in pre-existing cavities or by metasomatic processes, or both. The rôle played by the intrusive rock in this hypothesis would be merely to have furnished at its more or less shattered contact a plane of weakness favorable to the passage of meteoric solutions. The magnetite found as a considerable part of the ore-bodies would then have to be accounted for, as Read suggests, by the supposed action of subsequent reducing solutions.

The position of the ore-deposits, however, is adverse to this theory. We find them underlying an intrusive contact. After intrusion and all its after-effects had ceased, the contact was probably not an open one; nor is there anything in the nature of an impervious foot-wall to guide the solutions or to retard and induce the precipitation of their burdens. Then, too, we should have to appeal to further processes to account for the nature of the ore and its gangue-minerals.

There is no evidence, such, perhaps, as a basic rim to the granite laccolith, to suggest that the ore-bodies are possibly due to ultrabasic magmatic segregations; nor does the character of the ore lend support to such a theory.

The natural hypothesis is that the ores are due to the pneumatolytic after-effects, or, as Stevens[9] has recently amended the term, the emanation residues, of the granite intrusion. The ores have been expelled out of the cooling granitic magma in gaseous, vaporous, or liquid solutions, and have been dropped along its contact, probably in part filling open spaces, caused either by the shrinkage of the granite mass in cooling or else by the partial dissolution of the limestone, or both, and in part replacing the limestone by metasomatic processes.

Ore-deposits of this type of genesis have now been recognized for a number of years and have been described from many localities. Leith has classified them under the designation of "Pegmatite Type."[10] Especially does his description of the iron-ore deposits of the Iron Springs District, Utah,[11] present many analogies with the Ta-yeh deposits.

[9] Physical Data of Igneous Emanation, *Trans.*, xliii., 184 to 193 (1912).

[10] Iron Ores of Canada, *Economic Geology*, vol. iii., No. 4, p. 277 (June-July, 1908).

[11] *Bulletin No.* 338, *U. S. Geological Survey* (1908).

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The Ta-yeh iron-ore bodies are, in fact, a noteworthy example of this type of deposit, deserving of far more study than it was my opportunity to give to them. I feel that detailed study of this field should be rewarded by some valuable contributions to our knowledge of this interesting class of ore-bodies, while at the same time presenting some fascinating problems to the investigator.

The Concentration of Iron-Ores.

BY N. V. HANSELL, NEW YORK, N. Y.

(Cleveland Meeting, October, 1912.)

I. General.

The preparation of low-grade iron-ores by concentration, whether or not followed by an agglomeration of the concentrate, has in the United States only recently been recognized as a metallurgical process of economical importance for its iron industry. This importance of the process lies not only in the fact that it makes possible the use of ores heretofore discarded on account of their low iron-content or their too high percentage of elements deleterious in the iron industry, but also in its production of a raw material for the blast-furnace that is more uniform in its physical character and chemically purer, with a higher iron-content, than the general run of natural ores. When mixed in the burden of the furnace, it lowers the amount of coke and fluxing-stone required per ton of pig-iron produced. The furnace-operations become more regular and more easily controlled. The pig-iron is of higher quality and more uniform in character. In brief, it is becoming recognized that the beneficiating of the natural iron-ore, even if it makes the furnace-ore higher in cost per unit of iron entering into the burden, may mean a lowering of the production-cost of the pig-iron and an improved product.

At the present time, the Lake Superior district furnishes about 80 per cent. of all the iron-ore yearly produced in the United States. If the Southern States are exempted the percentage runs as high as 93 per cent. These ores are showing a steady decrease in grade from year to year. For the past 10 or 15 years this falling off in the average grade has been about 0.5 per cent. a year. The lowered grade of ore means an in-

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creased consumption of ore, fuel, and flux per ton of pig-iron, which in its turn causes an increase in labor, relining, and overhead costs (per ton of pig), due to decreased output per day. Writing of Pittsburg conditions as to grade of flux and coke, Edwin C. Eckel[1] says in a recent paper that a decrease of 5 per cent. in the iron-content of the ore involves the additional use of about 0.17 ton of ore, 0.12 ton of coke, and 0.36 ton of fluxing-stone per ton of pig-iron produced. Using these data and inserting approximate current prices for the three raw materials, it will be found that a drop in ore-grade from 50 to 45 per cent. brings with it an increase in cost, for raw materials alone, of about $1.25 per ton of pig-iron. When the other elements of cost, which also increase as the ore-grade lowers, are considered, the total increase becomes still more serious. Mr. Eckel draws the conclusion that it seems safe to assume that the cost of pig-iron will increase at the rate of between 25 and 50 cents per ton for each decrease of per cent. in the average iron-content of the ores to be

Considering these facts, notwithstanding [illegible] increasing tonnages of fairly high-grade ores that is being imported, it seems reasonable to presume that the demand for high-grade concentrates to be used as an intermixture with low-grade ores in preparing the furnace-burden soon will make itself apparent in a willingness on the part of the ore-consumers to pay the higher price that the concentrated product must command on account of the cost of its preparation beyond the mere mining-cost. The higher cost per unit of iron in the burden will be counterbalanced by the lowering of the cost of the furnace-operations, the increased output of the furnace, and the higher quality of the pig-iron produced.

This broader aspect of the problem gives us the right viewpoint from which to consider the advisability of purifying an iron-ore before charging it into the blast-furnace rather than during the smelting-operation itself.

The object of concentrating ores is generally to enrich their metallic content by reducing the percentage of gangue-minerals inherent in the natural ore. With iron-ores, this may not always be the chief purpose, as sometimes it is found advisable to concentrate ores with an iron-content sufficiently high for

[1] Iron Ore Reserves, *Engineering Magazine*, vol. xliii., No. 6, p. 835 (Sept., 1912).

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ordinary blast-furnace practice, the reason for the treatment being the necessity of reducing the percentage of elements generally objectionable for the iron industry, such as sulphur, phosphorus, copper, titanium, etc. The enrichment of the iron-content is then merely incidental.

Not all iron-ores are amenable to concentration in the present state of the art unless there are included under this general heading such metallurgical processes as the roasting of siderites or the nodulizing of Cuban limonites, both of which processes increase or concentrate the iron-content, and possibly also so-called metallizing processes—still in their experimental state—by which it has been claimed it is possible to handle those low-grade, close-grained, hard hematites of the Ranges with the silica so closely combined with the iron-mineral as to be significantly described as "silica poisoned," that to this date have defied all usual methods of concentration. Excluding in our present discussion such special processes, we may say that the basic principles of iron-ore concentration are well recognized and its limitations well known.

The various methods, then, which are at present commercially used in the United States in the concentration of iron-ores, can be classified as follows:

1. Washing of brown, clayey ores and sandy hematites, usually without previous crushing.
2. Wet-separation on jigs and tables of hard hematites, after crushing and sizing.
3. Dry-separation of hematites.
4. Magnetic separation of magnetites (and in a few cases mixed magnetites and hematites) by either dry or wet methods.

My intention in the present paper is not to describe in detail the different processes and the equipment used in the various plants in the United States, but to try to show more broadly the present status of the art and the possibilities for further development.

As can be concluded from the classification of the processes, the choice of method depends almost entirely on the character of the crude ore. The degree of refinement to which the preparation of the ore is carried in each separate case is influenced by such other factors as the comparative merchantable value of a higher or lower grade of product and its relation to the cost of production; the distance from, and the nature of, the

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market; the possibilities of selling the tailings-product or of obtaining from it some valuable by-product, like copper or apatite, or else the cost of its disposal; the handling of the water, if a wet method is selected, etc. The type and character of the ore is, however, the main consideration. In contemplating this, it is soon found that it is seldom that ores from different deposits are entirely alike in physical respects. A method found very successful for the beneficiating of one ore has to be considerably modified in order to give satisfaction in the treatment of another ore apparently quite similar.

A few remarks on the proper method of investigating the ore in connection with the determination of the method of enriching it are therefore pertinent before an attempt is made to describe the various methods as they have been worked out in the United States. Preceding this description of the difference between the processes, some attention will also be given to recent developments in ore-crushing, which enters as an important part of every installation of this kind, with the possible exception of washing processes for clayey ores.

As in every ore-concentration problem, the first step towards arriving at the best method to use is a careful investigation of the ore in the field and in the laboratory, followed by practical crushing- and concentrating-tests in the testing-plant. The general structure of the ore is studied, the mineral form of the metal-bearing constituents and the gangue-minerals is determined, as is also the size of the mineral grains, and the way they are attached to each other. Before a separation of the various grains can be made, they have to be freed from each other by crushing or otherwise, so that the difference in their physical qualities—specific gravity, magnetic permeability, etc.—can be taken advantage of in their separation. Valuable information for the determination of the most efficient and economical method for the comminution of the ore to the right degree of fineness for the separation can be procured in these preliminary investigations by a microscopic study of characteristic thin sections. It gives a correct knowledge of the mineralogical composition of the ore, which otherwise may be difficult to obtain in finely granulated and closely intermixed ore-samples. It will, for instance, give an explanation of the high iron-content in the tailings-product of an ore with gangue-

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minerals consisting of iron silicates high in iron. The percentage of iron in such minerals varies greatly in specimens from different localities, as investigations by Prof. W. Petersson[2] indicate. In amphibole, for instance, he found from 2.11 to 16.50, with an average in samples from nine sources of 8.76 per cent.; in pyroxene, from 3.05 to 12.00, with an average in six samples of 9.2 per cent.; in garnet, from 17.45 to 21.50 per cent., and so on.

Again, the study of the microsection may indicate that there is an appreciable amount of hematite intermixed with the magnetite, making the iron-recovery by magnetic separation unsatisfactory. It will show how the ilmenite and magnetite occur in a titaniferous ore and what may be the possibility of [illegible] rating them. It will clearly establish how the sulphur- [illegible] phosphorus-carrying minerals are intermixed with the rest [illegible] the aggregate, and thereby give data helpful in determining how fine to crush the ore in order to make possible their elimination in the separation.

Figs. 1, 2, and 3 show photographs of three such thin sections enlarged 16 times. In these the opaque or black parts are the magnetite; the more or less grayish, various iron silicates; and the white, chiefly quartz.

Fig. 1 is a coarsely crystalline gneiss carrying magnetite from an ore-occurrence in the northern Adirondacks, N. Y. The minerals are all in allotriomorphic grains, with quartz and pyroxene forming the largest part of the sample. The magnetite is in grains of all shapes and sizes, and forms, seemingly, the cementing-material between the rest of the aggregate. A crushing to $\frac{1}{8}$ in. will permit a clean separation.

Fig. 2 illustrates a bedded magnetite consisting of alternating layers with either quartz or magnetite predominating. The magnetite occurs in well-developed crystals, and, although the ore has to be ground fine for separation, it will give clean, high-grade products. The sample is from an eastern Mesabi deposit.

Fig. 3 is a sample of an amorphous hematite bedded alternately with micro-crystalline quartz. The sample is from eastern Vermillion and characteristic of several deposits there.

[2] *Jern-Kontorets Annaler*, vol. lviii., p. 359 (1903).

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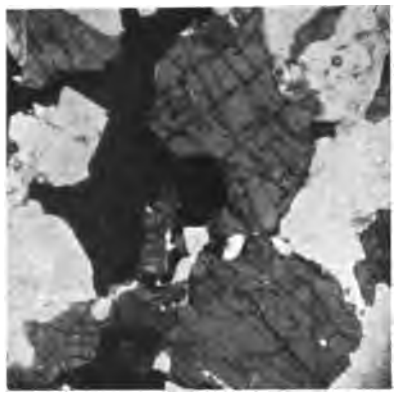

FIG. 1.—CRYSTALLINE GNEISS AND MAGNETITE.

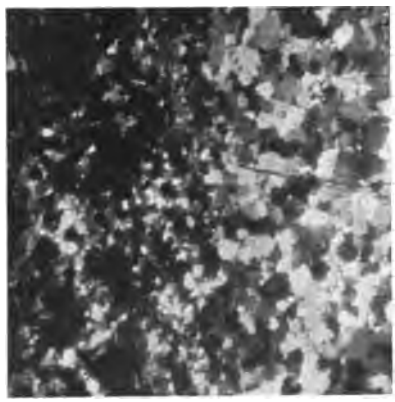

FIG. 2.—QUARTZ AND MAGNETITE.

FIG. 3.—QUARTZ AND HEMATITE.

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The hematite layers are practically solid hematite, while the quartz layers carry some diminutive hematite grains. A clean separation is next to impossible, as the intergrowth of the hematite and gangue precludes a clean crushing unless the grinding is carried to an exceedingly fine mesh.

This determination of the mineralogical composition of the ore, the size of the mineral grains, and how they are intermixed, gives a clue to the proper method for crushing and how far the comminution has to be carried; a most important matter on account of its being the most expensive part in the treatment of the ore.

II. Crushing.

The last few years have seen rapid strides in the improvement of crushing-machinery, and it must be admitted that rather than advancement in the design and construction of the separators, it is these improvements in methods and machinery for the crushing of the ores, and the resulting relatively low cost of their comminution, that have made it possible to handle profitably magnetic iron-ores containing even less than 30 per cent. of iron. These improvements in ore-crushing machinery are noticeable among both the primary coarse-crushing machines and the fine-grinding machinery.

Both jaw-crushers and gyratory crushers have been improved in design and constructional details. Cast-steel has replaced cast-iron in frames, and special steels in wearing-parts have prolonged their life, lowering the up-keep cost. Especially worthy of attention are the giant sizes which have made possible the use of steam-shovels in open-quarry work in the exploitation of the immense bodies of low-grade magnetites not uncommon in the eastern United States, in the Lake Superior district, and in eastern Canada. Opinions differ in regard to which of the two types of initial crushers mentioned—jaw-crusher or gyratory crusher—is preferable. In general, it may be stated that at present jaw-crushers apparently are more in favor when the desired capacity of the crusher does not exceed 300 tons per hr. One advantage is that less head-room is required; another, quicker repair in case of breakdown.

Edison giant rolls are less favored for use in iron-ore quarries, possibly on account of their capacity being so far above anything yet attempted in the mining of hard low-grade ore.

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They crush more than 1,000 tons per hr. The percentage of idle time would therefore be considerable, which would make the crushing-cost high, inasmuch as they consume a large amount of power, whether running light or operated to their full capacity.

Regardless of the type of crusher that has been selected as the initial crushing-apparatus of the plant, it is evident that it ought to be of sufficiently large size to take the largest piece that can be economically loaded and brought to it. It has opened the way for steam-shovel loading in open quarries, with its many contingent economies. Block-holing is eliminated. The stopes can be carried higher. The drilling- and blasting-expense is greatly reduced. It also creates a greater independence of labor-conditions by considerably decreasing the number of men needed in the quarry.

The rule is, then, that the first crusher must take any stone that the shovel can load. For a jaw-crusher this means a receiving opening of at least 48 by 60 in. in size. Some recent installations are using sizes ranging up to 60 by 84 in., an enormous machine that can take almost anything that the largest steam-shovel can lift into a car.

The preliminary crusher usually breaks the ore to a size of about 8 in. It is followed by secondary crushers of either type, screens, and other machinery; the crushing-plant being laid out as an ordinary stone-crushing plant with a gradual reduction of the ore to 1.5 in. To this point, the handling of all kinds of hard ore is alike, regardless of the method that has been selected for the subsequent treatment, based on the study of the ore itself and the results obtained in tests.

In this subsequent treatment, the concentration is generally designated as coarse or fine, two relative terms indicating whether the particles to be separated in the final stage of the process are coarse as sand or gravel or fine as slime. Whenever it is possible, coarse concentration is preferable, as it represents economy. As will be shown later, it is often combined with fine concentration, by which the middlings and tailings obtained by it are re-treated.

These general remarks appertain to the concentration of any low-grade iron-ore, although only more indirectly to the wash-

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ing of brown ores or sandy hematites, in which process a crushing of the crude ore is not generally required.

III. Washing.

The washing of brown ores and sandy hematites has been employed for many years in the South: in Georgia, Virginia, Alabama and elsewhere. Taking Georgia as an instance: the ore is found in the form of boulders and nodules in residual clays, and the proportion it bears to the entire material varies greatly. The concentrating process is simple and the plants in the South do not often involve any large outlay of money. A log-washer followed by shaking-screens or trommels, with picking-belts for the oversize and jigs for the fines, constitutes the usual equipment of the washing-plant. The slime is generally wasted without any attempt being made to recover its iron-content. The average iron-content of the concentrate is probably between 40 and 46 per cent., although at places it may run as high as 56 per cent. Generally, from 3 to 4 tons of wash material is handled for 1 ton of clean washed ore.

The peculiar development of the iron industry of the South makes the process well-suited to the conditions. The iron-ores are abundant and cheap, and the blast-furnaces are generally placed in close proximity to the mines. The ores often contain a considerable amount of lime with the silica, so that there is no great incentive to make a product of very high iron-content. The proper field for improvements in the process lies therefore in recovering part of the iron which now is wasted with the slime and the fines. It is reported that tests are being made with this point in view with reciprocating-tables. Experiments are also being carried on at several places with magnetic separators.

On the western Mesabi an experimental plant along similar lines was erected by the Oliver Iron Mining Co. in 1907, to determine the most economical way of treating the low-grade sandy hematites of the district. These lean ores occur as layers of hard hematite and free sand, alternating with paint-rock and taconite in varying amount. The results of the experiments were so satisfactory that a couple of years later a large permanent washery was built, which began working in the spring of 1910, and has since been in continuous operation. The plant

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consists of a number of parallel units, each with a capacity of about 200 tons of crude ore per hour. The principle of the treatment is the same as in the small washeries of the South, but the concentration is carried further, inasmuch as the overflow from the log-washers is concentrated on Overstrom tables for the recovery of a part of its iron-content. No crushing is done, because most of the large pieces are either good ore or taconite, which is picked out on the picking-belts. A similar plant has recently been put in operation by the Wisconsin Steel Co. at Nashwauk, Minn., to serve the Hawkins mine on the Mesabi range. It also is built on the "unit" system, each unit consisting of a grizzly with bars spaced 9 in. apart, a trommel with 2-in. perforations, a picking-belt for the oversize from the trommel, and two log-washers, for the undersize. The overflow from the log-washers is thickened in settling-tanks, and re-treated in auxiliary washers. No tables are employed. The present total capacity of the plant is about 4,000 tons daily.

The introduction of this system of washing ore on the Mesabi range is making it possible to utilize profitably the vast deposits of low-grade sandy iron-ore found in various parts of the range. This ore as it comes from the mines runs from 35 to 50 per cent. The iron-content of the concentrate is from 58 to 62 per cent.

IV. Jigging.

Low-grade specular hematites, in at least two places in Canada, are concentrated on jigs after having been crushed and screened along the usual lines employed in the wet method of concentrating ores by gravity. No attempt is made to produce a high-quality product, the object of the treatment being to reduce the silica and obtain an ore of a guaranteed iron-content with a minimum amount of fines. The crushing is therefore not carried further than to about 1.5-in. pieces. A drawback is the relatively large percentage of iron in the waste, which is unavoidable on account of there being no recrushing of jig-tailings. The recovery of this iron would require a fine crushing of the tailings, with a separation on sand-jigs and tables. The resultant product would be fine concentrates that would probably have to be agglomerated before shipment.

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The concentration of low-grade hematites and brown ores by jigging will undoubtedly be more used in the future. It is only relatively recently that large jigs have been constructed that can handle ores crushed only to 1.5 or 2 in. in size.

An interesting development in this connection is the Woodbury system of concentration, which is used extensively in the mills of the Lake Superior copper country, and which seems well-suited for the treatment of low-grade hematites. The ore is fed unsized to a plunger-jig so arranged that the flow of ore travels across the jig together with the water-currents caused by the plunger-impulse. The jig acts as a classifier, and delivers, besides coarse concentrates, hutch-sand, that may be re-treated on tables, middlings for regrinding, and slime, approximately 60-mesh and finer, for subsequent treatment on tables. The 24-in. classifying-jig has a stated capacity of 250 tons per 24 hr. of unclassified ore, crushed small enough to pass $\frac{3}{8}$-in. round hole. The 48-in. machine treats 500 tons per 24 hr. The classifying-jig is generally followed by several compartments of Woodbury jigs to complete the separation of concentrates, middlings, and tailings. The system greatly simplifies the lay-out of a mill.

The great capacity of the modern jigging-machinery is an illustration of the rule that nowadays things have to be done in a large scale in order to be done effectively and profitably.

V. Dry Concentration.

Before passing to the description of the magnetic concentration of iron-ores, a few words ought to be said about the dry process for the enrichment of "non-magnetic" iron-ores. There is, in the first place, the straight drying process as used at the Hollister mine in the Crystal Falls district, Mich., and at the Brunt mine on the Mesabi range, the purpose of which is a reduction in hygroscopic moisture of the ore, chiefly in order to lower the freight-charges. The Hollister ore assays 49 per cent. of iron, net. It contains from 13 to 14 per cent. of hygroscopic moisture. By the drying this moisture is lowered to 6 or 7 per cent. and the iron increased correspondingly.

There is also the E. G. Goltra process, which is developed on a commercial scale at Waukon, Iowa. Letters patent,

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recently published, by which the process is protected, describe it as follows: "The ore and its gangue are heated at a temperature and for a period sufficient to render the gangue brittle and easily separable from the ore. The ore and its gangue are tumbled to shake loose the gangue from the ore, and the ore and gangue are subjected to a strong air-blast so as to sweep, clean and remove the gangue from the ore." In regard to the practical application of the principle and the economical results obtained nothing has yet been published. It is known, however, that the drying- and calcining-furnace is similar to a nodulizing-furnace with the exhaust-fan at the charging-end. The discharged dry ore is screened. The oversize is cobbed by hand on a picking-belt. A part of the fines is concentrated on air-tables.

The Goltra process is probably best adapted to treat ores containing chiefly clayey gangue-constituents.

Air-tables have a limited use elsewhere in connection with the separation of dried finely-granulated hematites. They correspond to the shaking-tables used in the wet mills, and usually have a similar reciprocating, differential movement. The table-top is made of broad-cloth through which air is pressed either in a steady or a pulsating current.

At the briquetting-plant for flue-dust at Duquesne, Pa., such tables are used to re-treat the non-magnetic product from magnetic separators. They deliver hematite as concentrates and coke and lime-dust as tailings.

VI. Magnetic Separation.

Magnetic separation is applied to iron-ores which are naturally magnetic. It may also be profitably used for such ores as can cheaply be converted into a magnetic state.

The magnetite is strongly magnetic, *i.e.*, highly permeable to the lines of magnetic force. Introduced in a magnetic field it becomes itself a magnet, that is attracted to the magnetic poles from which the lines of force emanate. This causes the separation from the gangue-minerals, which as a rule are influenced only slightly, if at all, by the magnetic flux. The magnetic separation is simple and efficient and is the only method now used for the enrichment of low-grade magnetites.

The hematite is relatively weakly magnetic, *i.e.*, in a magnet-

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ic field it concentrates within itself but few lines of force. It can be separated from its gangue-minerals only by being exposed to a very strong and concentrated magnetic field. Machines designed for such weakly magnetic ores, although expensive in first cost and with rather small capacity when compared with separators for magnetite, will probably gradually be developed to commercial usefulness for hematites. What is particularly needed is a separator that can treat efficiently the overflow and slime from log-washers or hematites ground fine in water. It is probable that the evolution lies in this direction rather than in a conversion of the hematite into a magnetic oxide by heating in a reducing atmosphere.

Siderites give off carbonic acid when roasted at a temperature of only 300° C. and form a substance having the approximate composition of Fe_3O_4. They can then be separated as natural magnetic iron-ores. There is undoubtedly a field to cultivate in this direction in many parts of the United States.

The concentration of low-grade magnetites on magnetic separators is used at present to a relatively large extent for converting into a marketable blast-furnace ore the magnetic iron-ores of the East, which seldom occur sufficiently high in iron to be smelted in their crude state. A number of plants are in operation in the States of New York, New Jersey, Pennsylvania, and North Carolina, and in eastern Canada; some of them are well-known in their various details from recent descriptions in technical journals. The different methods employed at these works can advantageously be divided into the following three classes:

1. Coarse crushing with cobbing.
2. Fine crushing with dry separation.
3. Crushing with fine grinding and wet separation.

1. *Cobbing.*—If it has been found that the ore when broken to 1.5 or even to 0.5 in. in size contains a large percentage of reasonably pure magnetite, it can be separated into two products, of which the one may be a saleable furnace-ore. The other is either a waste product so low in its iron-content that it would not pay to fine-crush it in order to recover this iron or a low-grade material that is ground finer and concentrated.

This separation of coarsely-crushed ores is called "cobbing,"

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the magnetic cobber being substituted for the hand cobbing on picking-belts sometimes used for non-magnetic ores.

Cobbing-separators, therefore, like picking-belts, are often used in magnetic separating-plants as an intermediate step for reducing the amount of ore that has to be fine-ground. There are two ways by which this can be accomplished, *i.e.*, either by picking out, as mentioned, the rich ore-pieces by the use of a somewhat weak magnetic field in the cobbers or by the elimination of pieces of rock. In the latter case a stronger field is used so that all pieces containing sufficient iron to make them worth fine-grinding are recovered as the head-product. Sometimes, the plant is laid out with an initial cobber operated with low amperage, producing a rich furnace-ore. This is followed by a secondary machine with a stronger magnetic field by which pure rock pieces are discarded. The rest, being a middlings product, is reduced in size and re-treated.

2. *Dry or Wet Magnetic Separation.*—By the dry method, the aim is to produce as coarse a concentrate as is consistent with a clean separation and a satisfactory recovery. The procedure is therefore gradually to reduce the ore to the right degree of fineness. Each crushing-machine is followed by a screening-device so that undersized particles can be taken out and not subjected to further comminution. Drying the ore is required both on account of the difficulty of screening damp material and because dry separators give good results with clean products only when working on perfectly dry ore. Dampness of the ore causes the finely-crushed particles to stick together. Even with drying, it is not possible entirely to prevent non-magnetic dust from adhering to the magnetic particles, which tends somewhat to lower the iron-content of the concentrate or prevent an efficient reduction of phosphorus. This adhesion can be somewhat lessened by passing a strong current of air through the stream of concentrate as it is being discharged from the separator, by which non-magnetic dust is blown out and by means of an exhaust-fan conducted outside the mill. A close screening of the crushed ore and a feeding of the various sizes to different separators have been shown materially to improve results in dry separation.

As a rule, however, the dry method is particularly adapted to ores of coarsely crystalline character, such, for instance, as the

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Mineville ore of New York State. The several separating-plants in operation at Mineville are giving excellent technical and economical results. They constitute, in reality, the best examples of American practice in ore-separation, and serve as models for similar installations elsewhere. A great deal of experimental work has always been carried on there and the system has gradually been brought to a high state of simplicity and efficiency.

At present there are four mills in operation at Mineville, treating ore from the various shafts and deposits. As these mills form the largest dry-concentrating installation in the United States, with the latest and most approved equipment, an outline of the process as practiced in one of the mills will be given.

The No. 3 mill receives the ore from the two shafts of the Harmony mine. The ore is crushed in the two shaft-houses to 4-in. size. It is then delivered to a 300-ton bin, from which it is fed to an elevator. This discharges by means of a dividing-box to two revolving screens with 2-in. holes. The oversize is crushed in secondary crushers and returned until the whole flow of ore is reduced to 1.5-in. size. It then passes over two magnetic drum-cobbers, which remove the richer part as finished product. The rest continues over two cobbers of a different type with stronger field, which eliminate pieces too lean for further treatment; 12 per cent. of iron is considered the approximate limit. The heads from these cobbers are crushed in rolls to $\frac{1}{8}$-in. size and then delivered to Ball & Norton separators. The tailings product is crushed finer and re-separated. The concentrates from all the separators are collected in the shipping-bin and the final tailings are transported to the tailings-dump or sold for concrete-work or ballasting. The capacity of the mill is about 100 tons of crude ore per hour and the power-consumption is about 3 kw-hr. per ton of crude ore.

Analyses.

	For 1 ton of Concentrates.	Iron. Per Cent.	Phosphorus. Per Cent.
Crude ore	1.5	51	0.2
Concentrates	1	64	0.08
Tailings	0.5	10	

At Lyon Mountain, N. Y., is treated the ore generally known under the name of Chateaugay ore. The general system of

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concentrating is similar to that at Mineville. All the ore is crushed to 0.25 in. before being delivered to the separators, which are of the Ball & Norton double-drum type. Middlings and tailings are recrushed and re-separated.

Analyses.

	For 1 ton of Concentrates.	Iron. Per Cent.	Phosphorus. Per Cent.
Crude ore	2.6	32.0	0.022
Concentrates	1	64.0	0.009
Tailings	1.6	9.0	0.034

A screen-test of the concentrates made a few years ago with full analyses of all screenings has been plotted on the chart,[3] Fig. 4. It clearly shows how improvements in the system could be instituted. By the elimination of the coarse that stays on the 8-mesh sieve, and the very fine that passes through 100-mesh, of which there is no less than 15.6 per cent., the iron-content of the concentrates would be raised to more than 67 per cent. and the phosphorus reduced below 0.008 per cent. It shows that the ore ought to be ground so as to pass $\frac{1}{8}$-in. mesh, and it also shows that non-magnetic dust adheres to the fine concentrates, lowering the value of the whole product.

The wet method would unquestionably be technically superior. The whole mass of ore would be crushed in water to, say, 20-mesh. There would be no middlings and tailings to retreat, and no expense for drying the ore. The concentrates, washed free from non-magnetic particles, would show up better in both iron and phosphorus than by the present method.

For finely-granulated ores, such as are illustrated in the thin sections, Figs. 2 and 3, the wet method is probably indispensable. The ore has to be ground to 60-mesh before a separation of the magnetite grains from the gangue can be successfully attempted. Crushing and grinding the ore to such a degree of fineness produces a large amount of absolute fines. To separate these dry is out of the question. To crush dry is possible, but not advisable, as only with great difficulty can the fine dust produced in the operation be kept reasonably well inclosed. It will pass through the finest cracks. Even with strong exhaust-fans connected to casings around the various machines, the dust escapes, fills the atmosphere in the mill and makes the work in

[3] *Engineering Magazine*, vol. xxxviii., No. 4, p. 533 (Jan., 1910).

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the plant disagreeable and unhealthful, besides causing difficulties in the up-keep of the bearings, etc.

Grinding in wet ball-mills can be done cheaply, and at the same time permits a very simple layout of the plant. After the preliminary crushing to 1.5-in. size the ore is fed di-

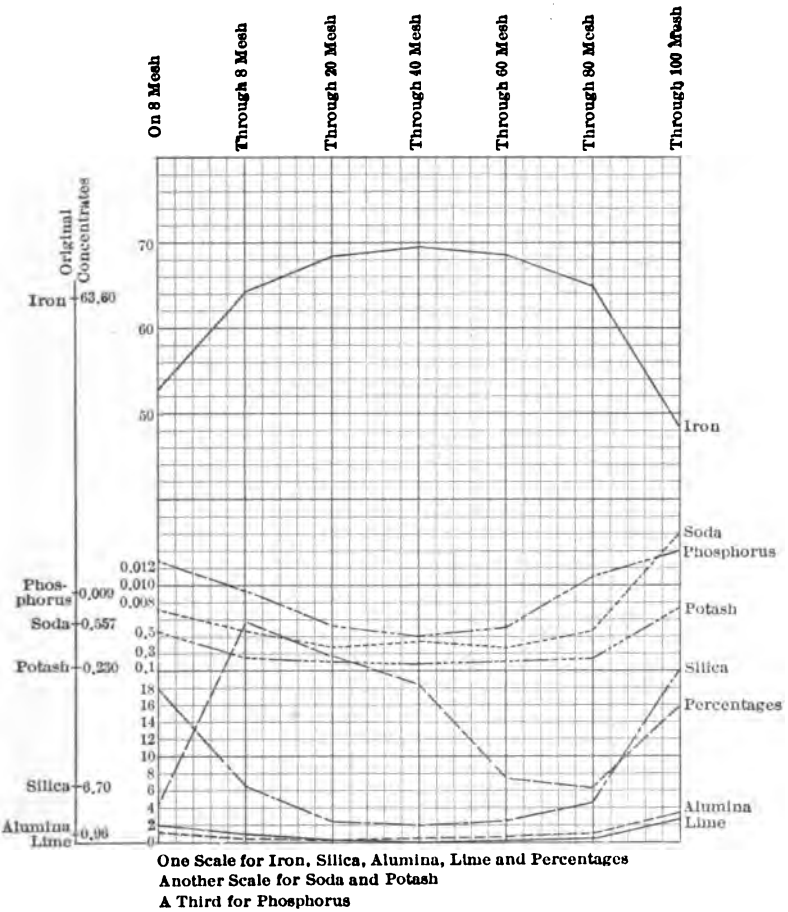

FIG. 4.—SCREEN-TEST OF IRON-ORE CONCENTRATES AT LYON MOUNTAIN, N. Y.

rectly to the ball-mill and in one operation ground to 20- or 30-mesh size in water. Driers, rolls, screens, and elevators are unnecessary. The ore, discharged from the ball-mill, is carried by a current of water directly to the separators. With some ores high-grade concentrates are here directly obtained. With other ores a finer grinding is necessary if high-grade

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concentrates are desired. The proper method, then, is to use after the ball-mills magnetic separators of such intensity that they will pick up the greater part of the iron mineral, leaving a tailings-product that can be wasted. The concentrate is crushed finer in pebble-mills and re-separated, yielding a product of any desired quality.

Among wet concentrating-plants for iron-ores in America are to be mentioned the one at Lebanon, Pa., operated by Pennsylvania Steel Co., on low-grade iron-ores from Cornwall Ore Banks, Pa., and the one at Sellwood, Ont., belonging to Moose Mountain, Ltd. At both places the Gröndal magnetic separators are employed. At the Moose Mountain plant, which is very modern in layout, having only recently been completed, the ore from the quarry is broken in a 36 by 48 jaw-crusher to about 3-in. size; it is then distributed to a four-unit plant, each unit consisting of one Jordan impact crusher, which is supposed to comminute the ore to less than 30-mesh, two Gröndal separators, one Hardinge conical pebble-mill, finishing-separators, and dewatering-machinery. The concentrate is formed into briquettes by a special type of briquetting-machine developed by Moose Mountain, Ltd., and is sintered in Gröndal briquetting-kilns.

Analyses.

	For 1 ton of Concentrates.	Iron. Per Cent.	Phosphorus. Per Cent.
Crude ore	2	36.19	0.072
Concentrates	1	65.58	0.020
Tailings	1	6.0	

In this short review of the present status of development of the magnetic concentration of iron-ores in the United States and Canada, only a few of the mines have been named at which separating-plants are in operation. The total capacity of all the magnetic separating-plants in America exceeds considerably 2,000,000 tons of concentrated ore yearly.

In certain parts of Europe, particularly in Sweden and Norway, magnetic ore-separation is used extensively. Norway prides itself on having the largest single plant of this kind. It is at the Sydvaranger mines in northern Norway. The ore there is similar to that of Moose Mountain. It is concentrated and briquetted by the Gröndal method. The present

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yearly capacity of the plant is 500,000 tons of concentrates, and it is now being enlarged to a capacity of 750,000 tons a year.

3. *Concentrates.*—The concentrates from the separating-process can be used directly in the blast-furnace without any agglomeration, unless the material is so fine that its use would cause inconvenience. In some cases, as at Lyon Mountain, N. Y., and Benson Mines, N. Y., the concentrates are classified either by screening or by being exposed to a strong air-current. The fines so produced are agglomerated in nodulizing-furnaces. The coarser material, which is supposed to be free from grains finer than 40-mesh, can be marketed and is used without any disadvantage.

For the concentrates from wet-separating plants, which generally are much finer than those from dry plants, agglomeration is usually required before the product can be marketed. There are three methods for the agglomeration of such fine ore that are used in America.

1. Nodulizing in revolving cylindrical kilns.
2. Blast-roasting by the Dwight-Lloyd or the Greenawalt process.
3. Briquetting by the Gröndal method.

In my paper [4] read at the New York meeting, February, 1912, a comparison between the various processes is given. They all convert the fine ore into a lumpy condition suitable for charging in blast-furnaces.

VII. Conclusions.

In making the first rough estimates on the probable revenue in the exploitation of a low-grade concentrating iron-ore, proper attention must necessarily be given to the tonnage of ore available and to the cost of mining it. Generally, any low-grade proposition requires the investment of a large capital in a concentrating-plant and in many cases in an agglomeration-plant, indispensable for the conversion of the fine concentrates into a marketable product. The operations have therefore to be laid out in a large scale, because otherwise the fixed charges distributed on the tonnage will make the cost of production seem prohibitive. A large tonnage of ore in sight, insuring operations on a large scale for a number of years, is therefore a prerequisite.

[4] The Briquetting of Iron-Ores, *Trans.*, xliii., 394 to 411 (1912).

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In the case of iron-ore, it is generally not difficult to predetermine the available supply of ore, and with magnetic ores it is even easier than with hematites, because the magnetic qualities of the ore make it convenient to employ some method of magneto-metrically surveying the ore-deposit. If correctly done, this will give the extent in field of the ore-deposit, together with useful data on the dip and pitch. By a few diamond-drill holes, where the magneto-metric map indicates the best location for them, the quality of the ore in depth and its continuity are determined.

With these preliminaries well done, the development of the deposit can be quite accurately planned. The cost of the various operations can be estimated very closely from figures obtained elsewhere in similar undertakings, and modified to suit the local conditions as experience dictates. The risk generally considered inherent to mining-enterprises is almost entirely absent. The consumption of iron-ores is large and as a rule increasing from year to year. The American production of pig-iron during the last 100 years has on the average somewhat more than doubled in each decade. The market is therefore well established. The market price of the ore has varied considerably and has, the last couple of years, been very low. It is, however, at present improving. In the light of the rapidly-growing consumption it is not assuming too much to believe that it will never reach a lower point than it has in the past season. The easily available ores of good quality are, as is well known, very limited in extent and are controlled by companies not desirous of exhausting them at too rapid a rate. The foreign ores of high grade are available chiefly on the coasts. Their transportation far inland would make their cost prohibitive.

By taking all these conditions into consideration, one can feel reasonably certain in calculations of the probable revenue in the development of a low-grade iron-ore proposition, using the present ore-prices in the various furnace-districts as a basis for finding the value of the concentrated and agglomerated product.

It is interesting to compare the various items of cost entering into the production of a high-grade blast-furnace material from a low-grade magnetic ore. We may take as an example

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an ore averaging 36 per cent. of iron in its crude state and occurring in such a large deposit that for several years to come it can be mined in open quarry and loaded with the steam-shovel. If the intention is to produce concentrates with 64 per cent. of iron, it will be necessary to mine 2 tons of crude ore to make 1 ton of finished product.

The plant is, we will say, laid out for the treatment of 2,000 tons of crude ore daily, which corresponds to 1,000 tons of concentrates in 24 hr. A plant of this tonnage can be considered of normal size. It is assumed that power can be obtained in the form of water-power, and that, transmitted to the works, it does not cost more than $25 per horse-power year.

Labor is calculated at 20 cents per hour. The quarry and coarse-crushing plant are operated only in the day-time because labor enters heavily especially into the quarry-operations, and therefore better economy is obtained by accumulating during the day-shift enough crushed ore, which is distributed in the bins over the fine-crushing plant, to last over night. The quarry- and coarse-crushing machinery is also easily obtainable in large units, making it convenient and economical to handle from 200 to 300 tons per hour.

In such a plant the following items of cost would enter into the cost of producing 1 ton of finished product.

Mining and transportation to crusher	$0.75
Crushing	0.10
Grinding and concentration	0.40
Agglomeration	0.50
Tailings handling and pumps	0.05
Shops, local administration, taxes, insurance	0.15
	$1.95

To this cost of manufacture is to be added the royalty on the ore as mined or as shipped, charges on account of the sinking-fund, selling-expenses, and freight to furnaces. These will probably bring the total estimate for the agglomerated product laid down at the furnace to $3.75 per ton.

As soon as the operations have reached the point where it is necessary to resort to underground mining, the cost of the product will be increased, but the calculations still show a profit in the operations on account of the high quality of the material, with its freedom from phosphorus and sulphur, which will make it bring a premium in the market.

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Discussion.

E. G. Spilsbury, New York, N. Y.:—I notice that Mr. Hansell has spoken in his paper about the work under the Goltra patents which is now being carried on at Waukon, Iowa. I had occasion a few months ago to visit that plant and was surprised to see what they were doing. The ore-body is practically a clay bed containing nodules of hematite of all sizes from large boulders down to small grains. It is mined by steam-shovel, and the iron content of the whole mass as mined varies from 22 to 26 per cent.

The result of the use of the Goltra process in the dry concentration of this ore is a material running from 55 to 59 per cent. of iron, and physically in a condition remarkably desirable for the blast-furnace. The process is rather a curious one; the ore delivered from the shovel is brought down to the mill, and after screening so as to take out the very large boulders, which go to the crusher, the whole mass is delivered into the horizontal revolving-kilns 155 ft. long and some 10 ft. in diameter. The secret of the success depends a great deal on the preliminary heating, which must be so graduated as to prevent the balling-up of the clay. The endeavor is to dry the material gradually in such a way that the clay is left in a finely divided state. The blast is introduced at the discharge end of the rotary kiln and actually blows out all of the clay-dust with a comparatively small percentage of iron. The ore as it progresses towards the discharge end is gradually raised to a very high heat and as it comes from the kiln is plunged into water and then runs through revolving-screens. The oversize from these screens is delivered to a long picking-belt. The process of roasting and plunging in water shows up the chert and quartz very thoroughly so that they are easily picked out. The fines passing through the screens then go to a second rotary kiln, which is hermetically sealed, and into which is admitted a certain amount of oil for fuel. The result of this second heating in a reducing atmosphere is the production of a magnetic oxide. This product then goes to the magnetic separators, which are of the Ball-Norton type.

While all the difficulties from the practical operation of the process had at the time of my visit not been entirely overcome, I consider that the process has the elements of very great suc-

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cess, and if the figures claimed for the cost of treatment do not exceed 30 cents per ton of concentrated ore, it ought to be a very profitable one.

At the time of my visit no treatment was being given to the flue-dust resulting from the blowing-out of the fine clay, but experiments had been made on a pneumatic concentrator on this material which had given very good results, and with such concentrators introduced, the element of loss of ore in the treatment will be reduced to a minimum. The process is certainly a very novel one and shows great ingenuity on the part of the inventor.

PHILIP N. MOORE, St. Louis, Mo.:—It is interesting to note the recurrence, with the use of better mechanical appliances, of an idea carried out years ago. In the 70's J. H. Hillman, recently deceased at Pittsburg, then superintendent of the Center Furnace of the Hillman Co., in western Kentucky, blew the siliceous dust from roasted brown ores by the use of a fan, thereby materially enriching them. These ores were roasted in open heaps, and, as the district has a considerable rainfall, the process was only of value during the dry season.

J. E. JOHNSON, JR., Ashland, Wis.:—What advantage has this roasting-process over the old-fashioned process of log washing which has been practiced in the eastern Pennsylvania and southern regions for many years, and with which I was at one time very familiar? We could carry that on and take out from the ore everything that was not attached to the ore itself, and we could do it for about 7 cents a ton; I think this process mentioned here would go closer to $1 a ton, and I do not see what is going to pay for the difference, or where there is room for much better recovery than we got, because we discarded very little. The loss in weight was about one-third, and in many cases that was so low in iron that it did not pay to treat it any further. While this process is interesting, I do not see where its commercial value comes in as compared with the old-fashioned method of washing.

MR. SPILSBURY:—They have been using the washing-process for a great many years, and obtained material which was probably from 46 to 48 per cent. of iron, whereas under this new

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method they are producing material of 66 per cent. iron content, and it is in a condition desirable for blast-furnaces.

MR. JOHNSON:—Most of that increase, however, came from the expulsion of the combined water, which is done automatically in the blast-furnace.

E. V. D'INVILLIERS, Philadelphia, Pa.:—I do not think that with the use of the old washing-method they have ever had anything as lean as 25 per cent. of iron to treat. Might there not be some economies in the washer to work materials of that kind. The ores were lean, I do not think that they were ever washed much better than from 42 to 46 per cent. of iron in the old days, with the attempt made to carry the process through jigging-methods to get rid of the free quartz.

MR. JOHNSON:—There are plenty of places in Alabama, Virginia, and Georgia, where they expect to handle 12 tons of dirt to get a ton of ore, and it is done right along as a commercial proposition. These ores are frequently raised to 46 per cent. of iron, and I know of a case in which the iron was raised to more than 50 per cent.

C. Q. PAYNE, New York (communication to the Secretary*):—Mr. Hansell has presented in his paper a very interesting and valuable summary of the various methods of enriching iron-ores which have been developed during recent years. Particularly striking in this connection is the high degree of enrichment of the concentrates from magnetic iron-ores which is now commercially feasible. Magnetic concentrates containing from 67 to 69 per cent. of metallic iron, showing an efficiency of 90 and even 95 per cent. in their separation, are not now uncommon. Such results exceed those obtained by gravity methods of treatment of all copper-, lead-, and zinc-ores, and are only equaled in the cyaniding of gold-ores and in the flotation treatment of certain copper-ores.

It is of interest to note that the type of magnetic field which is employed in securing these results, in the Gröndal (wet) and the Ball & Norton (dry) systems of treatment is the same in each case: viz., a field of alternating north and south poles.

* Received Jan. 22, 1913.

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The credit for developing this simple and useful field, which has found so wide an application both in this country and abroad, is due to two American inventors and engineers, Clinton M. Ball and Sheldon Norton. At the same time, it should be noted that part of the improvement in the grade of the magnetic concentrates, especially in the case of the wet treatment of finely mineralized ores, is due to a well-devised scheme of mill-treatment, whereby a rough concentrate is usually obtained as an intermediate product, which is then recrushed and reseparated in obtaining final clean products.

It is not unreasonable to expect further improvement in the design of the magnetic field over that of the Ball-Norton type may be made, especially in its application to the dry treatment of magnetic iron-ores, in order that with an equally careful mill-treatment the grade of dry concentrates may be brought to the same level of enrichment as those obtained in wet concentration.

Mr. Hansell's suggestion of the need of applying the concentrated or condensed-field type of magnetic separator to the wet treatment of those feebly-magnetic ores, like hematite and limonite, which in the aggregate offer so much larger a tonnage available for concentration than the magnetic iron-ores, opens up a very interesting field of inquiry. Having undertaken considerable work in this direction several years ago, I would like to mention some of the conclusions which I arrived at, in the hope that others may thereby avoid some of the shoals which seem to be amply provided for those who embark on this uncharted "sea of troubles."

In the first place, hematite particles cannot be so strongly magnetized in a condensed field as to inductively magnetize other hematite particles in their neighborhood, as is the case with magnetite particles. Theoretically, therefore, the sheet of ore fed into the field should be only one particle thick. This means that only a limited capacity of treatment may be expected from a relatively expensive machine. Moreover, those ores which have iron-bearing silicates among their gangue-minerals, or red colored clays, are difficult to enrich, since for practical purposes such silicates and clays are equally as magnetic in a condensed field as hematite and limonite ore particles, and they are therefore carried with them into the concentrates.

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These obstacles of small capacity and potentially imperfect results apply to both the wet and the dry treatment of hematite and limonite ore. In the wet treatment the more sluggish behavior of ore particles in water than in air, the greater difficulty of observing and controlling the separating action, owing to the muddy condition of the water, also the tendency to rusting of the pole-pieces, etc., from slightly acid water, are added difficulties to be encountered. The wide variation in the character of hematite iron-ores and their associated gangue-minerals, however, makes it possible that valuable economic results may still be obtained in some cases in spite of these difficulties. It seems unwise, therefore, to condemn the use of the condensed-field type of electro-magnet for the treatment of hematite or limonite ores, either in the dry or the wet condition, in advance of a careful study of each ore-occurrence with respect to its mineralization, association, and other peculiarities.

Personally, I am inclined to the belief that a combination of gravity and magnetic separation will, in the long run, secure the best technical and commercial results, in the enrichment of many low-grade hematite iron-ores, which cannot be treated so successfully by either system alone. Fairly-rich concentrates, with elimination of middlings products, can be more economically obtained on coarse ore sizes by gravity treatment than by means of magnetic separation. On the other hand, the tailings-losses in gravity separation, especially in the slimes and fine sizes of such ores, are quite high. In the treatment of fine ore sizes, magnetic separation accomplishes its best work. These facts, it seems to me, afford good reasons for joining the two methods of treatment together, in order to secure the most satisfactory results in the case of many such ores. In carrying out such a combination treatment, it is preferable, for most ores of the character mentioned, to give the finely-crushed material from middlings or tailings products, after a gravity roughing-treatment, a reducing roast, so as to magnetize the iron-ore particles present, and then to subject the roasted material to magnetic separation. The concentrates so produced should be of a very high grade, and the iron-losses in the final tailings quite low. A higher degree of concentration of the iron can be obtained by combining the gravity and the magnetic concentrates than by gravity separation alone. Such a combined treatment will require a minimum amount

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of briquetting or nodulizing of the concentrates, if any, and it should, therefore, show a low cost of treatment per ton of crude ore, together with a high recovery of the iron available in the ore.

In order to illustrate more clearly the possibilities of the suggested treatment, I have summarized below the results obtained in six Gröndal magnetic iron-ore plants, and have placed the results so obtained over the corresponding results in the large hematite iron-ore plant of the Oliver Iron Mining Co., where the ore is treated by gravity methods alone:

	Crude.	Concentrates.	Tailings.	Recovery.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Average of 6 magnetic magnetite-ore plants...Iron,	35.2	67.9	6.2	90.1
One gravity Mesabi hematite-ore plant.....Iron,	35.0	58.0	14.1	80.0

If we may assume that the less favorable results of the hematite-ore treatment are largely due to the difficulties of treating the fine material, and also, by way of illustration, that magnetic separation applied to this portion of the ore would permit the combined concentrates to be raised to 60 per cent. of iron, and the tailings reduced to 9 per cent. of iron, then it would be possible to secure 1 ton of such concentrates from 1.96 tons of the crude ore instead of 1 ton of the present grade of concentrates from 2.10 tons of the crude ore. The recovery would thus be increased to 87.4 per cent. Such assumed results, which may prove to be below rather than above actual possibilities, would show an increased yield of 1,191 tons of concentrates from the 35,000 tons of crude material treated in the mill per day; and, taking $2 per ton as the value of the concentrates at the mill, would provide an increase of nearly $2,500 per day in the gross value of the concentrates from the same amount of crude material.

The importance of these results will be realized when it is recalled that those iron-bearing deposits of the Lake Superior region alone which contain 35 per cent. and more of metallic iron have been estimated to contain 67,640,000,000 tons.[5] Whether these vast deposits can be considered in the light of ore-reserves will depend almost entirely upon the recovery

[5] Geology of the Lake Superior Region, *Monograph LII., U. S. Geological Survey*, p. 492 (1911).

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of the iron which can be secured from them in the form of concentrates, and upon the economy of treatment. Since the combined treatment suggested above holds out a reasonable hope in both of these directions, it clearly warrants careful consideration.

F. L. GRAMMER, Leesburg, Va. (communication to the Secretary*):—Mr. Hansell comes from a country where charcoal is the fuel, and fuel is therefore costly. The Swedes appreciate more than most Americans the value of ore-concentration, not only in fuel-economy, but also in its influence on the quality of metal produced. Neither in these particulars nor in our knowledge of the dressing of iron-ore do I think we can claim to be their equals.

A 60 per cent. ore is not merely twice, but more than four times, as valuable as an ore carrying 30 per cent. Failure to appreciate this fact explains many idle American stacks, erected in defiance of it. Next to the rule that the equipment-cost of a blast-furnace increases approximately as the square of the hearth-diameter, I know of no mathematical relations involved in smelting iron which are less appreciated than the disadvantages of lean ores.

Mr. Orn told me, when I was in Stockholm, that 2,500 patents on magnetic concentration had been taken out. He may have been joking as to the exact figure, but his remark indicated the great activity of inventors in this field. The magnetic separators seem to represent two main ideas, namely, wet and dry separation. The wet separators, such as the Gröndal, are applicable to ores requiring the greatest comminution by reason of the small particles of associated minerals or because of the occurrence of magnetite with a soft mineral likely to cake, as in those of Cornwall, Pa. Of the dry magnetic separators some are well suited for large-grained ores, such as those of Mineville, N. Y. We have also magnetic separators, like the Wetherill, for ores not usually classed as magnetic.

A symposium on magnetic separation was held at Plattsburg in 1892, but the Mineville operations continued for years to be our sole successful commercial example of the magnetic concentration of iron-ore. Aside from skillful manage-

* Received Jan. 29, 1913.

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ment, this enterprise had the advantage of ores not requiring excessive pulverization—which is the item of greatest cost. Moreover, rejected material, apatite, furnished a revenue as a fertilizer. Now, at many places the tailings from such a concentration of iron-ore, if siliceous, can be used for concrete; if very fine, as "chicken-gravel;" if pyritic, for the manufacture of sulphur; if cupriferous, for copper-smelting; and if titaniferous, for electrical purposes.

In reading about the treatment of iron-ore to-day, one recalls that before Mesabi ores were introduced to any extent, say in 1892, one seldom heard in the Pittsburg region any reference to a sieve-test, save for the examination of Benson ore-concentrates, which were on market for a short time, or for "purple ore," a name by which the Natrona pyritic residues were then known.

Our attitude towards iron-ores has changed in another way. Then the best ores were abundant; and, having the freedom of choice, we were prompted by our limited metallurgical knowledge to select ores low in phosphorus, sulphur, titanium, silica, and alumina, and high in iron. It was a prophylactic method. Now we have entered on therapeutics; and being no longer able to avoid evils, must try to cure them. Prophylaxis is always best, but it is not always practicable. We cannot always command such superlative natural ores as those of Minas Geraes, Brazil; Imataca, Venezuela; Dannemora, Sweden, or Pewabic.

Nor do we now limit our thoughts so exclusively to the chemical aspect, and under that head we do not always regard only the elements regarded by laymen as "poisons," such as phosphorus, sulphur, antimony, etc. We are beginning to appreciate that there are, so to say, "spices," such as nickel and vanadium, which benefit quality, and which will be more highly valued and keenly sought in future.

The reasons for this change in our estimate of the value of iron-ores may be briefly summarized as below.

1. It is not a question of shortage in aggregate ore-reserves. Mr. Leith's estimate of 10,000,000,000 tons, published in 1908 in the *Scientific American*, has been greatly enlarged by Cuban and Brazilian deposits since reported; and Dr. Vogt, of Christiania, Norway, thinks (I believe correctly) that the world is not

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threatened with an early iron-ore famine. It is true that Lake Superior shipments from 1854 to the end of 1912 foot up about 576,000,000 gross tons, or say about 560,000,000 in 30 years; but Minnesota reserves are estimated in the last tax reports as 200,000,000 tons greater than five years earlier, showing, if these returns are reliable, that shipments have not equaled the amounts developed by prospecting. On the other hand, Michigan, which shipped 219,000,000 tons in the past, is now credited with reserves of 169,000,000 only.

What we have to face, therefore, is the depletion of older lump-ore regions, and the fact that while the iron-content of Lake Superior shipments from 1889 to 1898, inclusive, averaged 56 per cent. of iron, this average from 1899 to 1906, inclusive, was only 52.55 per cent. In this production of leaner ores we find our first explanation of increased interest in concentration. The great drain on Lake ores has been mainly in recent years, as the reports of the U. S. Geological Survey on Mineral Resources show.

2. Magnetic processes have greatly advanced in efficiency and economy.

3. Concentrates are more likely to be uniformly rich than the output of small mines feeding a few furnaces. I have known the blowing-in of two additional furnaces, drawign upon a given mine for their ore, to result in the reduction of the ore-shipments for a year from 48.5 to 44.5 per cent. of iron.

4. Blast-furnace managers, having had a taste of lean ores, now know more clearly what the use of such ores means in coke-consumption and irregular metal, and are asking for concentrated ores.

5. Some people have been aroused by the idea of "conservation"—a word introduced in forestry reform. It is unquestionably true that some corporations, to make a good showing of ore-reserves, have shipped leaner ore than formerly. This is a temporary expedient, which may be termed "conservation," and may make a good showing as regards the mining department, taken by itself; but it involves great extravagance in fuel-cost and fixed charges to the smelter, as already frequently shown. There is such a thing as an ore too rich for exclusive use in coke-furnaces; but one need not hesitate to use 62 per cent. iron-ores; and the richness of an ore, no matter

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how great, is a recognized element of value. It can always be sold at a high price per unit, to enrich lean mixtures.

6. Concentration not only means economy in fuel and fixed charges, but the running of a furnace on uniform ores permits close fuel-margins or low silicon. These influence very markedly large pig-iron outputs (a relation seldom adequately understood), and also lessen the time and the loss of steel involved in conversion.

7. Concentration means the elimination of impurities as well as the enrichment of the ore to a uniform quality. It renders available ores which are well located economically, but which would otherwise be unfit; and thus it unquestionably increases the practical ore-reserves of the world. In fact, the word "concentration" used by Mr. Hansell is not comprehensive enough to cover the mechanical preparation of ores, including crushing, purifying, enriching, and briquetting or agglomerating.

In this wider sense it is concerned not only with breaking lumps, securing pieces uniform in size and contents, and eliminating undesirable constituents, but also with their porosity, shape, and the texture, and their form of oxygen combination: all of which affect their reduction or their action in the open-hearth.

8. The attempt of the pyrites-burners to get a better price for their residues, the excessive amount of fine, as compared with hard, ores in the large reserves of the Mesabi range, and the desire to use the fine residual argillaceous hydrated light ores of Cuba, are also influences explaining recent American interest in this topic of ore-melioration or uplift.

R. H. Richards, Boston, Mass. (communication to the Secretary *):—The paper entitled The Concentration of Iron-Ores, by N. V. Hansell, is very complete indeed and leaves little for me to comment upon. I can, however, make a few remarks in regard to some experiences of my own.

In regard to testing iron-ores previous to making the first large-scale test, I have been able to gain very valuable information, which pointed to the direction in which the large-scale tests would be made, by the use of a 400-lb. sample of the ore, using a Blake breaker and rolls for crushing, a set of screens for sizing, a little hand jig 12 in. square, and another 6

* Received Feb. 17, 1913.

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in. square, for answering the questions about jigging, a hindered settling classifier with closed spigots, and a little Wilfley table 51 in. long and 26 in. wide for treating fine material; also a little Ball-Norton magnet belt mounted with 4-in. belt and a rheostat for giving a variable current from zero up to the maximum, and a Wetherill magnet with rheostat giving current from zero up, the maximum of the Wetherill magnet being about 80,000 ampere turns, and a little roasting furnace for making the ore magnetic before treatment. With this little testing plant one can answer most of the questions as to the correct size, the degree of sizing and classification, the per cent. of iron that may be expected in the concentrate and in the tailings, and the efficiency, or per cent. of extraction of the material, that may be hoped for; also the degree of reduction in quantity from the crude ore to the finished concentrate. These facts being obtained in regard to the ore, the first large-scale trial can be planned with a good deal more assurance of success than if such a test had not been made previously.

In regard to the pulsator jig, as a very compact and large capacity machine, we have been a little while in developing this, as it has taken time to discover the various needs of the design and the methods of running. We believe that all the difficulties have been overcome at the present time. There is just now a trial going on in the Marquette region which promises good results. The subject on which it is worked is the separation of iron-ore from quartz and jasperite. The experiment has not gone quite far enough yet to publish any description or figures of results.

Recent work on a jig at Lake Superior in the copper region is worthy of note. A single compartment pulsator jig with a screen 12 in. square has treated successfully half the output of the steam stamp, ranging from 300 to 350 tons in 24 hours, taking out all but the fine copper. The size of the crude ore is $\frac{5}{8}$ in. and under, and the tailings of this jig are believed to be sufficiently well cleaned from the coarse copper to go to a screen having 0.25-in. round holes, which delivers an oversize ready for recrushing, the undersize going down to the classifiers and jigs in the usual Lake Superior system. The treatment of this large quantity by so small a jig fully bears out the claims that have been made for the pulsator jig as to the great capacity of which it is capable.

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Surveying and Sampling Diamond-Drill Holes.

BY E. E. WHITE, ISHPEMING, MICH.

(Cleveland Meeting, October, 1912.)

In August, 1911, I read a paper before the Lake Superior Mining Institute[1] on surveying and sampling diamond-drill holes. The present paper gives a more thorough description of these methods, together with a few changes and notes suggested by further experience. Since August, 1911, the methods, used constantly by the Cleveland-Cliffs Iron Co., have been developed to a higher degree of efficiency, and have given great satisfaction.

Until a few years ago but little attention was paid to the determination of the actual course of drill-holes, to the accurate sampling and analysis of the material, or to the scientific location of the drill-holes. It has now been common practice for several years in many districts to test the inclination of drill-holes by etching glass tubes with hydrofluoric acid, but the direction of deviation from the vertical has not often been determined. Methods of determining the direction by means of a compass were first developed in connection with drilling in Victoria. The latest method used on the Rand in South Africa has been recently described by John I. Hoffman.[2]

In the same paper Mr. Hoffman describes an ingenious method of wedging-off a drill-hole so as to start new holes at several successive points. By this means a hole can be deflected in any desired direction, and the curvature can be controlled to a considerable extent, provided the rock is soft enough to permit the hole to be wedged off. Unfortunately, in the Lake Superior district, the jasper iron-formation is so hard that it is very difficult, if not impossible, to do this. I tried to deflect one hole so as to avoid drill-rods stuck in the hole, by using a

[1] *Proceedings of the Lake Superior Mining Institute*, vol. xvi., pp. 100 to 120 (1911).

[2] Recent Practice in Diamond-Drilling and Borehole Surveying, *Bulletin No.* 91, *Institution of Mining and Metallurgy* (Apr. 11, 1912).

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12-ft. wedge of the hardest steel obtainable, but the bit cut the wedge and hardly touched the jasper. If there had been a dike of softer rock higher up in the hole I might have been able to deflect it at that point. Two years ago I described[3] how the curvature of a diamond-drill hole can be controlled by other means, and gave an example where it had been done successfully. Although the hole in this instance kept to the desired inclination, it failed to intersect the ore encountered in a previous hole because the direction of deviation from the vertical of the previous hole had not been determined, although the amount of deviation had been determined by the usual hydrofluoric acid tests.

Since 1910, I have experimented with two methods of determining the direction of deviation, one applicable to non-magnetic and one to magnetic rock-formations, with very good results. We now test all drill-holes except very shallow ones for both the amount and the direction of deviation, usually at 200-ft. intervals, and we have the satisfaction of knowing very closely the actual position of any ore or other formation encountered. In non-magnetic rock-formations the old method of a compass suspended in gelatine is successfully used, with improvements worked out by George Maas and myself, and patented by Mr. Maas (Patent No. 1,003,624, Sept. 19, 1911). In magnetic formations a method of marking the drill-rods is used in connection with hydrofluoric acid tests. This method was, to the best of my knowledge, first applied by John Deacon, Superintendent of the Republic Iron & Steel Co.'s properties at Negaunee, Mich., who used it several years ago in testing diamond-drill holes at the Cambria mine. I found that with great care good results can be obtained by this method, but it is more difficult and expensive than the compass method. It is the only one I know of that is practicable in magnetic formations where the direction of strike underground is unknown. If the strike were known it would be easy to take an impression of the bottom of the hole with a piece of lead and determine the direction of deviation by comparing the direction shown by an acid etching with the strike shown by the piece of core broken from the bottom just before making the test, when made to coin-

[3] *Engineering and Mining Journal*, vol. xc., No. 12, pp. 546, 547 (Sept. 17, 1910).

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cide with the impression in the lead. Jean Florin [4] describes a similar method, using a compass and camera instead of the acid.

Surveying in Non-Magnetic Formations.

Figs. 1 and 2 show the cases used to test for inclination and course, using the latter when more than one test is to be made at one time, since it may be inserted at any point in the drill-

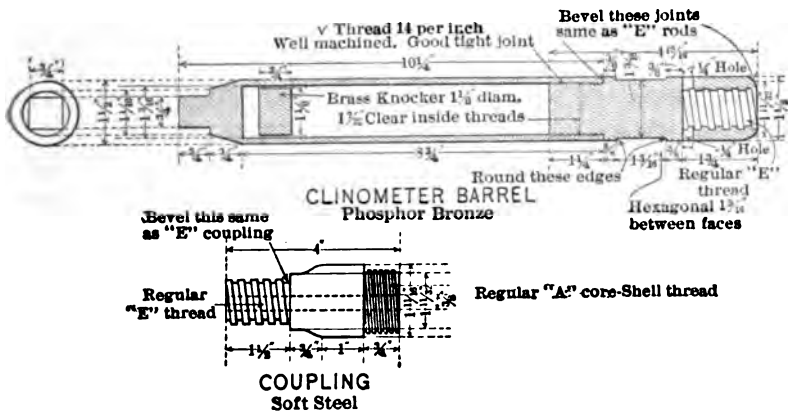

Fig. 1.

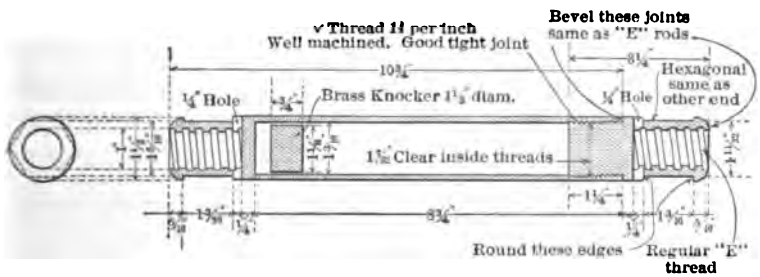

Fig. 2.

FIGS. 1 AND 2.—PHOSPHOR-BRONZE CASES TO TEST INCLINATION AND COURSE OF DRILL-HOLES.

rods at the same time that the first case is used at the end of the rods. As it is desired to use as large a glass tube in the case as possible, and as the outside diameter is limited by the size of an E hole, a material was selected which combined the greatest possible toughness and tensile strength with non-mag-

[4] *Annales des Mines de Belgique*, summarized in *Engineering and Mining Journal*, vol. lxxxvii., No. 17, p. 854 (Apr. 24, 1909).

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netic properties. Phosphor-bronze was chosen, which is entirely non-magnetic, and which can be obtained with a tensile strength of 60,000 lb. and an elastic limit of 54,000 lb. per sq. in. By using a case of the dimensions given in Fig. 1, a glass tube $1\frac{1}{8}$ in. in outside diameter can be used, and according to Nystrom's formula for the collapsing-strength of small tubes, $P = \frac{4Tt^2}{fd\sqrt{L}}$, using a factor of safety of 4, this case should be safe in a hole 2,800 ft. below water-level. A little wicking is used to make a perfectly tight joint.

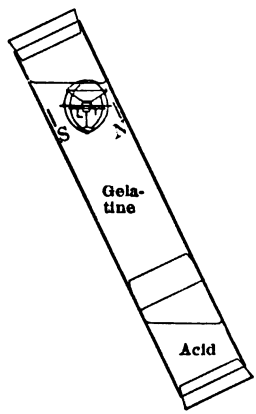

Fig. 3.

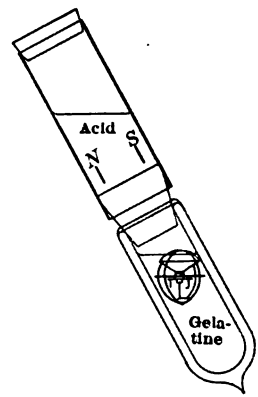

Fig. 4.

FIGS. 3 AND 4.—MASS PATENT DRILL-HOLE COMPASS IN TUBES, AFTER MAKING TESTS.

The compass invented by Mr. Maas, shown in Fig. 3, has the advantage over the old forms of compass used for this purpose that it is pivoted in a cage which prevents it from coming in contact with the glass tube, and insures a free swing in the gelatine. The cage is below and rigidly attached to a cork float.

The most accurate and satisfactory method of testing the course of a hole is to use the compass in a glass tube about 6 in. long, open at each end. A section of rubber stopple is forced into the tube, leaving about 1.5 in. space for acid at one end, and 4 in. space for gelatine at the other. The gelatine may be prepared beforehand, but usually a small weighed portion of dry gelatine is carried to the drill and dissolved on the ground in a given quantity of water, care being taken that the water has no chance to evaporate while dissolving the gelatine. The

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proportions are so chosen that when dissolved the solution will keep liquid as long as possible after being lowered in the drill-hole, and yet will become perfectly solid when cold. For instance, with Nelson's Improved Brilliant Gelatine we use $\frac{5}{8}$ g. and dissolve it in 50 cc. of water. In a hole where the rods can be lowered in 20 min. or less, a $1\frac{1}{8}$-in. tube is used with paper wrapping. When it takes from 20 to 30 min. to lower the rods, a 1-in. tube is used with several wrappings of paper. If deeper than this, a thermos bottle is used, and by wrapping with paper the gelatine may thus be kept liquid 50 min. By using 6 g. of pure common salt with the 50 cc. of water the gelatine may be kept liquid for 90 min. The time the gelatine remains liquid was determined by tests in water cooled by ice to 43° F., the approximate temperature of the underground water in the Lake Superior district.

In the first two cases, when the thermos bottle is not necessary, the dissolved gelatine is poured into the tube, which is then heated by immersing in water heated to boiling by live steam. When hot, the compass is dropped in and a stopple is placed in that end; then about 1 in. of dilute hydrofluoric acid is poured into the other end and that end closed. The tube is then wrapped in paper and placed with gelatine end up in the bronze case, which is attached to the bottom of 20 ft. of brass E rods and lowered into the hole, losing as little time as possible. The brass rods are screwed to the bottom of the regular drill-rods, using an A to E reducing-coupling if the hole is being drilled with A rods. The bronze case and brass rods are made for an E hole, so that they can be used in either case. If two tests are to be made at the same time, another tube and compass are placed in the case shown in Fig. 2 and inserted in the drill-rods at the proper point, using 20 ft. of brass rods on each side.

The tube is left stationary in the hole 50 min. after the rods are lowered, giving the gelatine time to cool and set, and the acid time to etch a good line. If salt is used in the gelatine the tube must be left stationary for 5 hr. after the gelatine starts to cool. Acid diluted with 12 parts of water gives the best results. This solution is prepared in the office and carried to the drill in hard-rubber bottles, which are much more convenient than the paraffin bottles in which the acid is usually furnished.

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When the tube is brought to the surface, the positions of the north and south points of the needle are marked on the glass with a diamond point and the tube is washed out. This tube then forms a permanent record of the inclination and course of the hole at the depth at which the test was made.

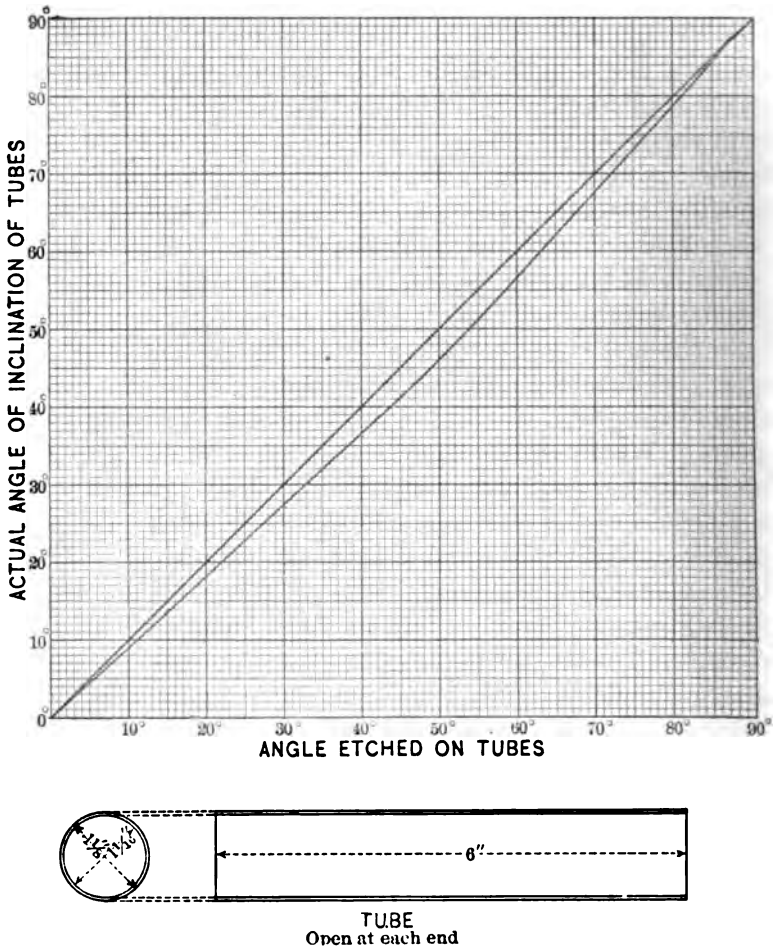

FIG. 5.—CURVE OF CORRECTION FOR CAPILLARITY IN TESTING INCLINATION OF DRILL-HOLES. RULED IN INCHES AND TENTHS.

The thermos bottle, 1⅛ in. in outside diameter, consists of two clear glass walls, with a vacuum between. When it is necessary to use this, both the hot gelatine and the compass are placed in the bottle, which is then closed by a rubber stopple. The stopple also closes one end of a 1⅛-in. tube 3 in. long, serving to connect the bottle and tube and preserve them

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in the same relative position shown in Fig. 4. Dilute acid is placed in the tube, the other end closed, and the tube and bottle placed in the bronze case and lowered into the drill-hole. It only takes the gelatine 90 min. to solidify in the thermos bottle, so that if it takes 40 min. or more to lower the rods, the test is usually left in the hole 50 min. after the rods reach the *bottom*, just long enough to get a good etching. It may be left

FIG. 6.—TWO-CIRCLE GONIOMETER TO MEASURE INCLINATION AND DIRECTION OF DRILL-HOLES.

in the hole over night, but in that case the acid should be more dilute. When the tube is brought to the surface, the north and south points are marked on it, corresponding to the position of the compass-needle in the thermos bottle. The 3-in. tube then forms a permanent record of course and inclination, just as the 6-in. tube does.

In either case the inclination is read in a goniometer and is

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corrected for capillarity according to a curve which is prepared for each size of tube by testing tubes at known angles. Fig. 5 shows a curve for 1⅛-in. tubes. It will be noted that for these tubes the correction is only 3.75° at 45°, which is the maximum. The angle can be read to 0.5°, and I feel certain that the results of tests for inclination can be relied upon to within 1°.

To determine the course of the hole, the tube is placed in a special two-circle goniometer, Fig. 6, with both circles set at 0°, so that the tube is vertical, and the vertical circle parallel to the crosspiece over the top of the instrument. If the inclination of the hole is steep, the tube is twisted until the etching shows the dip to be either directly towards or away from the eye; that is, until the cross-thread bisects the ellipse etched on the glass. If the inclination is shallow, it is more accurate to twist the tube so that the dip is to the right or left of the observer and in the plane of the vertical circle. The vertical circle, holding the glass tube, is then turned until the north and south marks on the tube are in line with the cross-piece, when a pointer on the horizontal circle reads the course of the drill-hole. Figs. 3 and 4 show tubes with acid, gelatine, compass, and north and south points marked, just as they are taken from the drill-hole.

We have found the method described above very successful, and two tests at the same point almost always agree to within a few degrees. When this is not the case, more tests are made, and so far we have always been able to ascertain which are correct. We have made tests at a depth of 2,000 ft., and by using salt in the gelatine, tests can be made at even greater depths.

The precautions to be taken are: 1, that the compass swings perfectly freely, and does not catch on the cage; 2, that the gelatine keeps liquid long enough; 3, that the compass is not used when there is much local magnetic attraction in the rock-formation; and 4, that the compass is not affected by the steel drill-rods or casing, or by other iron in the hole. The first and second precautions are easily taken; the third can be judged only by a knowledge of the formation and by taking tests at different depths, which, if concordant, would indicate that there is no appreciable magnetic attraction. The fourth precaution is important. We use two 10-ft. lengths of brass rods, and so have no iron within 20 ft. of the compass. Tests with

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10, 20, 30, and 40 ft. of brass rods at the same depth gave the same reading in a hole dipping 50° N. 45° E., so that a length of 20 ft. is conservative. The results of a second test with shorter lengths of brass rods at another point gave:

Length of Brass Rods. Feet.	Inclination. Degrees.	Apparent Course. Degrees.	Length of Brass Rods. Feet.	Inclination. Degrees.	Apparent Course. Degrees.
1	62	N. 6 E.	7	62	N. 41 E.
2	.....	N. 21 E.	8	.....	N. 23 E.
3	.....	N. 39 E.	10	.....	N. 48 E.
4	.....	N. 35 E.	12	.....	N. 50 E.
5	.....	N. 42 E.	20	.....	N. 47 E.
6	.....	N. 42 E.			

These data show that the true course is between N. 47° E. and N. 50° E., and that 20 ft. of brass rods is conservative. It is impossible to obtain results accurate to closer than 2° or 3°, because of the small size of compass necessary and the personal error in marking the glass tube and reading the goniometer.

Surveying in Magnetic Formations.

When the rock-formation is known to be magnetic, or when several tests with the compass do not agree, there seems to be no way of determining the course of a hole but by lowering the rods in such a way that the test-tube can be oriented at any point in the hole. We have done this by the method used by Mr. Deacon. The rods are first screwed together in one or two long lines on the surface, just as they will be lowered into the hole, with a bronze case at the end, and at intermediate points in the rods if desired, all the joints being made as tight as usual. Great care is necessary that no twist be left in the rods when screwing them together on the ground. This trouble is not experienced when there is snow, as the rods slip easily, and no torsion is introduced. When the ground is bare, difficulty may be avoided by placing level planks at short intervals for the line of rods to rest upon, and to prevent their touching the ground at any point. A straight stretch of railroad-track, with ties exposed, is very convenient for this purpose. If not more than 500 ft. long, the rods will turn on grass without leaving any twist in them.

When all connected, each joint, including the bronze case, is

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marked with a chisel, so that it can be screwed up again to exactly the same place. The joints are marked exactly on top as they lie on the ground, so that when the rods are in the hole the marks will point in the same direction. The rods are then broken into 20-ft. lengths, convenient for handling, being careful to keep one wrench on the couplings so as not to disturb any joints not to be broken.

Dilute hydrofluoric acid is poured into a glass tube, the tube marked with a diamond, and placed in the bronze case so that the mark on the tube corresponds with that on the case. A convenient way of doing this is to cut a mark across the top of the stopple, set this to agree with the mark on the tube, and then place the tube in the case so that the mark on the stopple agrees with that on the case. The tube is then lowered into the hole, being careful exactly to match the marks at every joint, and again keeping the wrench on the couplings. The mark on the last rod is placed directly in front of the drill and this direction determined, which is the direction of the mark on the glass tube. The tube is left stationary at the bottom of the hole for about 50 min. when acid diluted 12 to 1 is used, and then withdrawn and washed. To determine the course of the hole, another mark, exactly opposite, is made on the tube, and the course is found by using the goniometer as described in connection with the gelatine test.

This method, of course, gives accurate results only when the rods turn easily in the hole, so that there is no twist in them after they are lowered, which is usually the case except in very deep holes, or in holes where the inclination is low, or where the curvature is excessive. In these latter cases, unless the hole is rifled, the twist may probably be removed after the rods are in the hole by raising and lowering them several feet a few times. Precaution should be taken that the tube cannot turn in the bronze case, by forcing it in with a little waste alongside. On withdrawing the test it is well to examine each joint in the rods as they come out of the hole to see that none have moved while lowering or hoisting, and to be sure that the tube has not moved in the case.

This method requires great care and patience, but is capable of very good results. My experience has been that it cannot safely be left to assistants, whereas the compass method will

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give good results in the hands of any ordinarily careful and intelligent assistant, or even in the hands of the drillmen.

I have tested the same drill-hole by both methods and found the results to agree very closely. In one case the first tests in an old hole, by compass, did not agree very well because of casing left in the hole. Two or three tests were taken at each point, however, and the averages gave a smooth curve for the drill-hole. Later tests were made with marked rods, and the results agreed so well with the average of the compass-readings that when the two were plotted there was a difference of only 12 ft. at the bottom of the hole, although it was 1,697 ft. below and 262 ft. horizontally from the collar. The averages given below show that the hole described a spiral of considerable curvature. These results would have agreed even better if the compass had not been considerably affected by iron casing in the hole to a depth of 1,500 feet.

Depth. Feet.	Inclination. Degrees.	Course. By Compass. Degrees.	Course. By Marked Rods. Degrees.
100	89	N. 73 E.	N. 85 E.
300	87.75	S. 42 E.	S. 36 E.
500	86	S. 24 E.	S. 2 E.
700	86.5	S. 38 W.	S. 30 W.
900	85	S. 25 W.	S. 47 W.
1,100	79	S. 51 W.	S. 52 W.
1,300	71.5	S. 77 W.	S. 81 W.
1,500	65.75	N. 85 W.	N. 86 W.
1,700	64	S. 77 W.	No test.

Deflections of Drill-Holes.

J. S. Curtis[5] gives an interesting theory of the cause of bore-hole deflections, with results of experiments which he made to substantiate his theory. He endeavors to show that the influence of terrestrial magnetism should cause vertical drill-holes to deviate to the north in the southern hemisphere, and affirms that this is the case in the great majority of holes, although the direction may be changed by the character of the country-rock.

In our experience the latter feature is much the more important, and from results of our drilling I would not say that the great majority of drill-holes deviate either north or south in all

[5] *South African Mining Journal*, vol. viii., part 1, No. 379, p. 425 (June 11, 1910), and *Journal of the South African Institute of Mining Engineers*, vol. ix., No. 8, pp. 199 to 206 (Mar., 1911).

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districts. If the strata are flat and uniform the holes may do so, but if the strata dip steeply this is not the case. In one district, where the dip is steep, we are certain of the course of 14 holes which deviated from the vertical. Of these holes, one went approximately north, one approximately south, one NE., five NW., one SE., and five SW. Putting it in another way, seven deviated to the north and seven to the south, while two deviated to the east and ten to the west. If these results show anything, they only show that the majority of the holes deviated to the west, but equally to the NW. and SW.

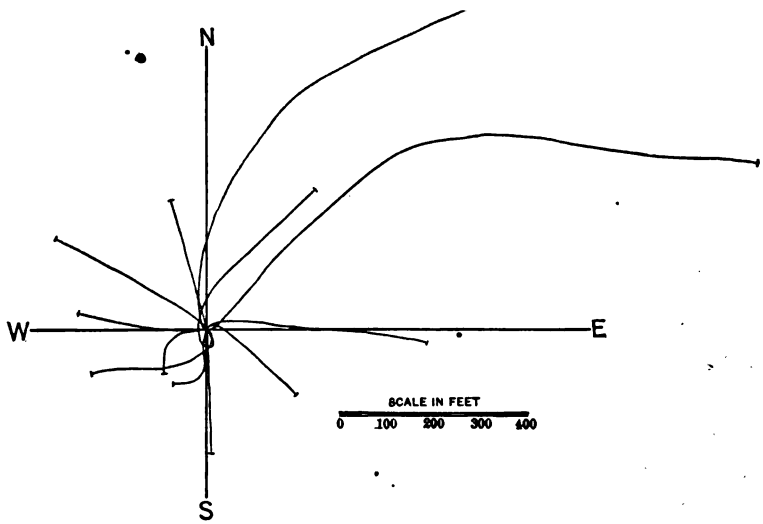

FIG. 7.—PLAN OF 12 DRILL-HOLES IN LAKE SUPERIOR DISTRICT.

Fig. 7 shows a plan of holes which deviated considerably from the vertical. These holes were drilled on several ranges in the Lake Superior district. It will be seen that there is no general course that can be predicted.

It is very difficult to keep vertical diamond-drill holes straight, and I believe that a hole can be located with more assurance of striking a certain point in depth if it is given an inclination of 85° against a steeply-dipping formation than if it is started in a vertical position. We have drilled only two holes with this inclination, the results being:

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Depth. Feet.	No. 1. Degrees.	No. 2. Degrees.
At surface.	85	85.5
At ledge.	86	86.5
200	86	.....
400	85	87.75
500	87.5	.....
600	87	.....
800	.....	88.5
1,200	.....	85.5

These two holes are not enough for a generalization, but they kept straighter than vertical holes in the same district. In addition, we knew in what general direction the holes would deviate, which we do not know when we start a vertical hole. In 1910[6] I gave a series of curves showing the curvature to be expected in an inclined hole when dipping against a steep jasper formation. I would change the curve for a hole started at 85°, since, under these conditions, that angle seems to be a critical one, and the hole does not flatten as would be expected.

In view of the sometimes surprising curvature of drill-holes, I feel that all holes should be tested both for course and for inclination at 100- or 200-ft. intervals, whether started vertical or at an angle, otherwise there is no certainty as to the exact place at which the ore or other strata is actually cut. We recently started a vertical hole which, at a depth of 800 ft., was found to have an inclination of only 51° from the horizontal. Another hole, started at an angle N. 54° W., was found to be running N. 64° E. at the bottom. A third hole, started vertical, deviated 377 ft. in a total depth of 1,290 ft., and was only 47° from the horizontal at the bottom.

Daily Reports.

Fig. 8 shows a report form filled out by the drillmen every shift for the drill-foreman and the head office, and Fig. 9 a sample-tag which is placed in every bag of core or sludge. The record of time of drilling and footage of each diamond-bit is kept, to obtain data on the several stones in the bit with the idea of determining which are the most economical. These tests have shown that the wearing quality of the stone depends considerably upon the specific gravity and upon the structure.

[6] *Engineering and Mining Journal*, vol. xc., No. 12, pp. 546, 547 (Sept. 17, 1910).

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THE CLEVELAND-CLIFFS IRON COMPANY,
Ishpeming, Michigan.
RECORD OF DIAMOND DRILLING.

..... Shift. ..... 191.....

Where Working : .....

Section : ..... Hole No. .....

	Feet	Core Saved	Hours
Total depth of hole per last report, - - -			
Moving and setting up, - - - - - -			
Drove ..... inch stand pipe, - - - -			
Drilled with chopping bit, - - - - -			
Drilled with diamonds, - - - - -			
Reamed from ..... to ..... ft			
Lowered ..... inch casing to ..... ft			
Total Depth of Hole, - - - -			

Kind of Material :			

	Feet	Size	Hours
Bit No. ..... - - - - -			
Bit No. ..... - - - - -			
Bit No ..... - - - - -			
Number of Men :			

REMARKS : .....

.....

.....

.....

.....

.....

..... Runner.

..... Helper.

..... Setter.

Report delays, accidents, etc. ..... Foreman.

FIG. 8.—REPORT FORM FOR RECORD OF DIAMOND-DRILLING.
(Original size, 5 by 8.5 in.)

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District.....

Sec..... Hole No. .....

Depth,.....

FIG. 9.—FORM OF TAG FOR SAMPLES.
(Original size, 2.5 by 3.5 in.)

Although it is important to have the drillmen report the amount of core saved from each material, yet they rarely measure it accurately, and if the analyses of core and sludge are to be combined, as described below, the core is remeasured when it reaches the office.

Samples from Drill-Holes.

The following directions for saving samples are posted in the drill-shanties and enforced by the inspector:

Directions for Saving Sludge from Diamond-Drill Holes.

Set the standard sludge box just below the floor of the shanty and in such position that there is room to siphon off the water and take out the sample without moving the box. Connect a tee to the top of the standpipe or casing and lead a pipe from it to the nearer end of the sludge box, at such a height that it will either be level or slant towards the sludge box and just rest upon the top of the box, and of such a length that it will not project more than one inch beyond the edge of the box. The pipe must not be more than two feet long, and if longer than one foot must be split on top for the foot nearest the sludge box, so that if sludge collects in the pipe it may be seen. Set the box level, so that water will overflow evenly across the whole width at the far end, and wedge the partition firmly, so that it is in close contact with the bottom of the box. The top of the partition should be one inch below the water level. The box is now ready to receive the sample, and drilling may be started.

While drilling, care must be taken that no water from the drill-hole escapes around or over the tee except through the pipe leading to the sludge box. Care must also be taken that there is no leak from the box and that the three-inch plug at the end of the box is tight. Sludge samples must be taken for every five feet drilled, or less, preferably from even five-foot intervals; that is, from 460 to 465, 465 to 470, 470 to 475, etc.

Whenever a sludge sample is to be taken, drilling must be stopped and the hole washed out clean. The pipe leading to the sludge box must be cleaned out into the sludge box, and either the pump must then be stopped or the tee turned so that the water will not be discharged into the box. Carefully remove the partition in the box, so as not to stir up the sludge any more than necessary, and when the sludge is settled, siphon off the surplus water, being careful to keep the end of the siphon near the surface of the water, and not disturb or draw off any of the

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fine sludge at the bottom of the box. To use the siphon, fill with water a three-foot length of large size flexible hose, and with one hand on each end place one end beneath the surface of the water in the box and the other end on the ground eight inches or more below the top of the box. When both ends of the hose are released the water will flow out of the box, and may be allowed to flow until it is seen that the sludge is beginning to go off with the water. Then remove the hose and thoroughly mix the sludge in the box to a mud. This must all be removed from the box and placed in a pan on the boiler to dry. The pan must be at least 8 in. by 12 in. by 1 in. deep, with flat bottom, and must be thoroughly cleaned each time before a sample is put in it to dry. If enough water cannot be drawn off without disturbing the sludge so that the sample can be contained in this pan, a larger pan must be used. All the sludge must be saved and the sludge box cleaned thoroughly. When the sludge has been cleaned out, remove the three-inch plug at the end of the box, and wash out the box with a pail or two of water, then replace the plug and partition, and drilling may be started again. The sludge must be labeled, giving the depths between which the sample was taken, when it is placed on the boiler to dry. It must all be saved and turned over to the inspector. Sludge must always be saved when drilling in iron formation or in any other ferruginous or red material. While drilling in material from which a sludge sample should be saved, if the water is lost, if the sludge does not come up with the water, or if the sludge is contaminated with material caving from higher up in the hole, drilling must be stopped immediately until the hole is put in such condition that good sludge samples can again be obtained, or until the inspector gives orders that drilling may proceed.

Whenever the drill runs into or out of ore, provided the band of ore or rock is one foot or more thick, drilling must be stopped, and the sludge box cleaned out immediately, without waiting to complete the five-foot run. When the drill runs out of ore continue taking and saving sludge samples for at least twenty feet, no matter what the material, so that it may be determined whether the ore is caving.

Keep the core separate from the sludge, and each time core is pulled, label it with the depths between which it was recovered. Each run of core must be kept separate, and all core must be saved and turned over to the inspector. When the core is pulled, if it is found that more core is saved than the proportion of one foot of core to ten feet of drilling, the sludge box must be cleaned without waiting to complete the five-foot run, and the sludge labeled and saved separately. If sludge from a shorter distance than five feet is in the box at the end of the shift's work, and if less than the above proportion of core is saved, the sludge may be left in the box, provided the shanty is locked and the box is inaccessible from outside the shanty. If anybody can get at the box, however, and if there is no watchman, the sludge must be removed from the box, dried, labeled, and placed with the other samples.

The standard sludge-box is shown in Fig. 10.

When the samples reach the office they are carefully examined, and a daily report of all drilling made out on the form shown in Fig. 11. Samples of all core and of all sludge which runs above 40 per cent. of iron are preserved in special cabinets in a fire-proof room. A few pieces of core are saved from each run, and the rest sent to the laboratory for analysis if the ore-

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formation has been cut, otherwise it is thrown away. A little of each sludge-sample is placed in a small pasteboard tray, with a temporary label, 10 ft. to a tray, until the analysis is completed. Each 10 ft. of sludge-sample which exceeds 40 per cent. of iron is then placed in a gelatine case 3 in. long by 1 in. in diameter, and preserved in the same drawer with the core. Gummed paper labels are used for both core and sludge.

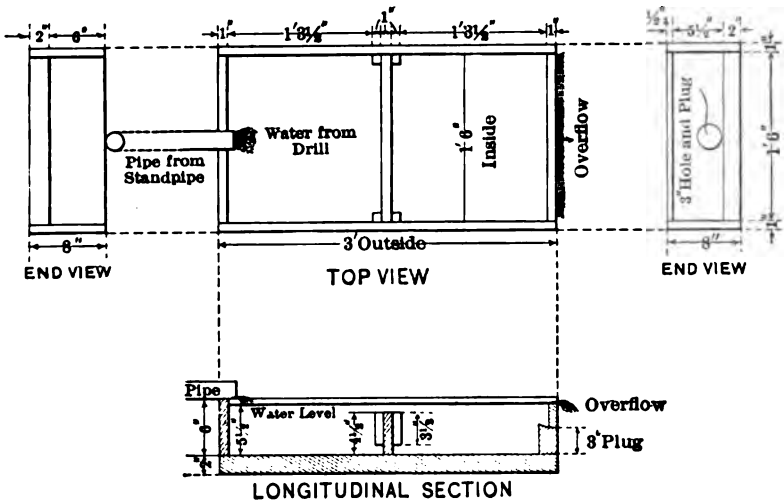

FIG. 10.—CLEVELAND-CLIFFS IRON CO. STANDARD SLUDGE-BOX.

THE CLEVELAND-CLIFFS IRON CO.

DAILY REPORT OF DIAMOND DRILLS.

Ishpeming, Mich., 191

Section	Hole No.	Date	Feet Drilled	Total Depth.	Material	Remarks

FIG. 11.—FORM FOR DAILY REPORT OF DIAMOND-DRILLS.

(Original size, 8.5 by 11 in.)

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Combination of Core- and Sludge-Analyses.

When enough core is saved to make it worth while, and when the analyses of the core or sludge exceed 45 per cent. of iron, the results are combined according to a formula which gives proper weight to each, derived as follows:

Let A = diameter of bit outside of carbon, in inches.
B = diameter of bit inside of carbon, in inches.
C = feet of core saved in "D" feet drilled.
D = feet drilled.
S = volume of rock actually ground to sludge, in cubic inches.
T = volume of rock actually saved as core, in cubic inches.

$$\text{Then } S = 12\,D\left(\frac{\pi A^2}{4} - \frac{\pi B^2}{4}\right) + 12\left(D - C\right)\frac{\pi B^2}{4}$$

$$\text{and } T = 12C\,\frac{\pi B^2}{4}$$

$$\text{Hence } \frac{S}{T} = \frac{D(A^2 - B^2) + (D - C)B^2}{CB^2} = \frac{DA^2}{CB^2} - 1$$

	Dimensions of Bits. Inside of Carbon.	Outside of Carbon.
Standard A bit	1 in.	$1\frac{1}{16}$ in.
Standard E bit	$\frac{27}{32}$ in.	$1\frac{9}{16}$ in.

$$\text{Hence for an A bit, } \frac{S}{T} = \frac{1.813^2 D}{C} - 1 = 3.29\,\frac{D}{C} - 1$$

$$\text{and for an E bit, } \frac{S}{T} = \frac{1.563^2 D}{0.844^2 C} - 1 = 3.43\,\frac{D}{C} - 1$$

To obtain an average of the sludge- and core-analyses, giving the proper weight to each, the sludge-analysis should evidently be multiplied by S, and the core-analysis by T, and the sum of the products divided by $S + T$. The result is the same and the operation simpler to multiply the sludge by $\frac{S}{T}$ and the core by 1, and divide by $\frac{S}{T} + 1$; hence, the rule for an A bit is to multiply the sludge-analysis by $3.3\,\frac{D}{C} - 1$, add the core-analysis, and divide the sum by $3.3\,\frac{D}{C}$.

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Only iron is run on the 5-ft. samples, with phosphorus if they contain more than 40 per cent. of iron, and manganese if exceeding 1 or 2 per cent. Occasionally, several of the 5-ft. samples are combined, and a complete analysis made for iron, phosphorus, silica, lime, magnesia, alumina, manganese, sulphur, titanium, and loss by ignition.

After combining the core- and sludge-analyses, the results are further averaged in continuous runs of ore of the same grade. We call from 45 to 50 per cent. of iron "lean ore," 50 to 57 per cent. "second-class ore," and more than 57 per cent. "first-class ore."

Representation of Results.

As soon as possible after the first of the month, the record of material drilled during the previous month is compiled from daily reports and averaged analyses. This result is carefully checked with the core and then recorded permanently in the drill-book, which has loose leaves, with pages of the form shown in Figs. 12 and 13. The reports of the drillmen and analyses of samples are copied in this book daily, and it forms the complete and permanent record of drilling. From this book tracings are plotted from which blue-prints are made. Figs. 14 and 15 show the printed forms on tracing-cloth used for this purpose. The tracings are 14 by 16 in. in size, the same as that of the loose-leaf sheets in the drill-book, and are bound in covers of the same size.

Cross-Sections of Drilling.

For the scientific location of drill-holes, cross-sections must be made through outcrops and previous drill-holes, preferably at right angles to the strike of the formation, showing the holes plotted according to the surveys for inclination and course, and showing the material encountered. We make these on cross-section tracing-cloth ruled in inches and tenths, on a scale of 50 ft. to 1 in. The sections may be superposed and compared, and the cross-section ruling makes it easy to read distances and areas without a scale. On these tracings the geological boundaries of formations and the outlines of ore-bodies are drawn in soft pencil, which prints satisfactorily, but may easily be erased and changed if further drilling shows the first

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assumptions to be wrong. When necessary, longitudinal sections are made, which, in connection with a plan and cross-sections, give a very good idea of the structure of the deposit.

THE CLEVELAND-CLIFFS IRON CO.

RECORD OF DIAMOND DRILLING.

District .....
Mine _____ *State* ..... *T.* ..... *R.* ..... *Sec.* ..... *Hole No.* ___ *Location:* .....

Dip: ..... *Course:* ..... *Drilled by whom* ..... *with* .....

Date	No. Hours	Core Recovered Ft.	Core Recovered In.	Stand Pipe	Drilled	Total Depth	Kind of Material	Begins at	Dip of Strata	Remarks

FIG. 12.—LOOSE-LEAF PAGE FOR RECORDS OF DIAMOND-DRILLING.

(Original size, 14 by 16 in.)

THE CLEVELAND-CLIFFS IRON CO.

District .....
Mine .....
Option _____
Lease .....

ANALYSIS OF CUTTINGS AND CORE.

State ..... *T.* ..... *R.* ..... *Sec.* ..... *Hole No.* ..... *Location:* .....

Determined by ..... 191 .... *Elevation of Collar* .....

From	To	Iron	Phos.

FIG. 13.—LOOSE-LEAF PAGE FOR RECORD OF ANALYSES.

(Original size, 14 by 16 in.)

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THE CLEVELAND-CLIFFS IRON CO.

_____ EXPLORATION.

Section_____ Township_____ North Range_____ West_____

☐ Test Pits.
O Stand Pipes.
⊕ Diamond Drill Holes.
◔ Diamond Drill Holes showing Lean Ore. 45-50%.
◒ Diamond Drill Holes showing second class Ore. 50-57%.
● Diamond Drill Holes showing Ore above 57%.

Scale 1″ = 600′.

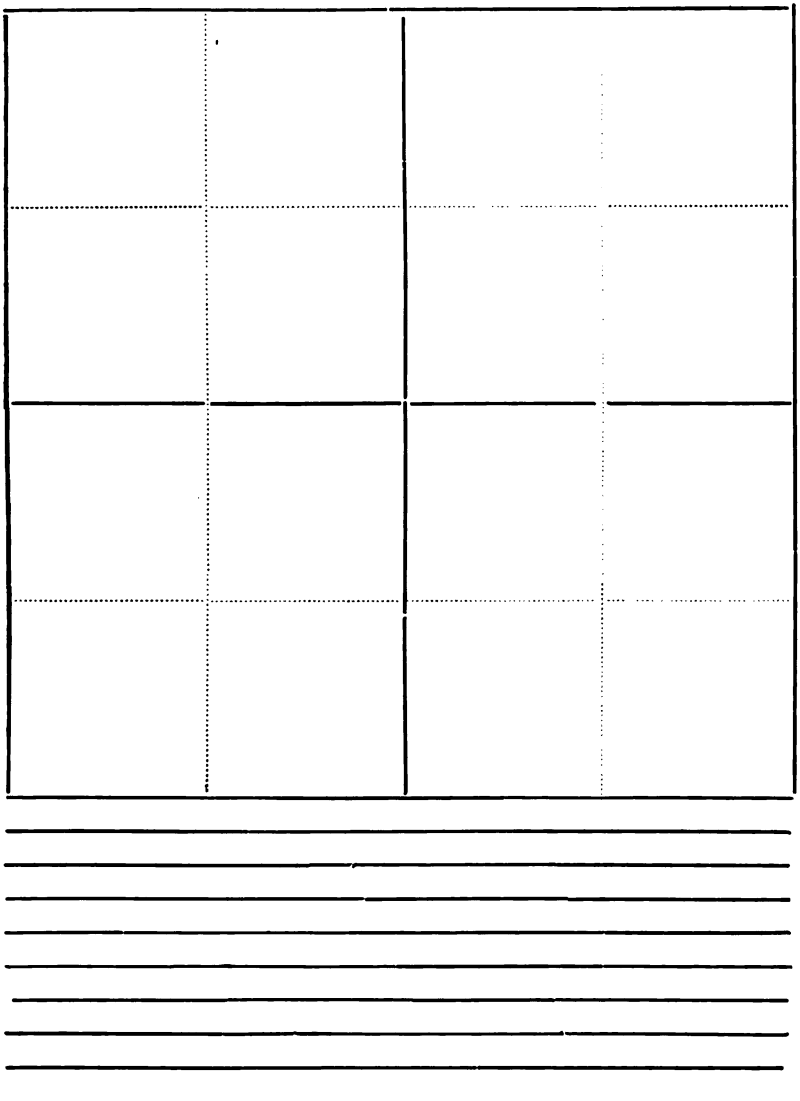

FIG. 14.—LOOSE-LEAF TRACING-CLOTH FORM FOR PLAN OF EXPLORATIONS.
(Original size, 14 by 16 in.)

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THE CLEVELAND-CLIFFS IRON CO.

_____ EXPLORATION DIAMOND DRILL HOLES.

Section_____ Township_____ North Range_____ West _____

Scale 1″ = 50′.

FIG. 15.—LOOSE-LEAF TRACING-CLOTH FORM FOR SECTIONS OF DRILL-HOLES.
(Original size, 14 by 16 in.)

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New Type of Blast-Furnace Construction.

BY J. E. JOHNSON, JR., ASHLAND, WIS.

(Cleveland Meeting, October, 1912.)

THE general construction of blast-furnaces has undergone no radical change in more than a generation. When the old style of masonry construction was replaced by the steel shell, the masonry piers were simultaneously replaced by columns, never less than 6 and frequently 12 or 16 in number, set under the mantle-ring. These columns were at first unvaryingly made of cast-iron, but in more recent years have frequently been made of structural steel.

The furnace itself has recently been the subject of radical changes; in some cases the thin-lined construction has been adopted for the whole furnace, and in other cases it has been adopted for a zone immediately above the bosh, raising the mantle several feet to make this possible, but still the style of construction with columns set immediately under the mantle has been universally followed. In spite of the apparent fixity of this type of construction it is open to grave disadvantages from the operating point of view, which may be briefly outlined as follows:

The slope of the bosh and size of the crucible are such as to leave scant room inside the columns for the necessary water-piping, etc. The bustle-pipe must necessarily be outside the columns and the penstocks have to pass between them, with the result that the columns are frequently much in the way when changing the tuyeres or otherwise handling the penstocks. Moreover, the hearth-jacket requires a diameter but little smaller than the circle of the inside of the columns, and as a result the cooling-water ditch for protection against break-outs is exceedingly limited in width. As a consequence of these conditions, when any work requires to be done around the furnace, such as changing a tuyere or working on the cooling-water pipes, the amount of room left between the columns, pen-

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stocks, and water-pipes is more conspicuous by its absence than by its presence. Moreover, if a break-out occurs, the ditch is filled with iron, perhaps half way around the furnace, or more. This locks itself in between the columns and the hearth-jacket in a way that not infrequently makes impossible its removal while in blast. I knew one furnace-man who declared that the proper thing to do with the ditch was to run the first cast of iron into it and let it stay there so as to save future bother.

Another result that not infrequently has followed break-outs is the burning and cracking of the columns by the molten iron and cinder surrounding them.

Still another result of this construction is less immediately disastrous; but leads eventually to a very serious condition. The space between the base of the columns and the hearth-jacket being so small, the whole structure is necessarily set on one foundation, and the continual expansion, which all masonry structures undergo from continued heating, gradually pushes the column-bases out. In some cases an effort has been made to resist this tendency by making a continuous sole-plate, or ring, on which the bases of the column rested, but I have never known a case in which these were not broken, and some plants have abandoned their use altogether, using simply an individual base under each column, on the ground that as they will break apart anyway, it makes a neater job to make them separate in the first place.

I think all furnace-men have experienced these conditions, and will agree to the general correctness of these objections to this construction.

It had seemed to me for several years that it would be possible to avoid these evils by building a frame-work of structural material strong enough to carry the weight of the whole furnace, and supporting this on columns set at the corners of this frame-work, which would throw them so far back from the furnace proper that they would be safe from all danger from break-outs or other accidents, and at the same time would allow room enough around the furnace for necessary access to all parts with great safety, ease, and speed in all required work; and that if the ditch became filled with iron, this would be unable

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to lock itself around the columns, and, being free on one side, could readily be removed.

When our investigation concerning the quality of charcoal-iron had indicated the desirability of reconstructing the furnace to obtain the lines needed, we laid out various plans for doing the work, and this construction appeared to be so practicable from an engineering standpoint, and so desirable from an operating standpoint, that it was finally adopted.

We first laid it out with the idea of using a triangular frame of girders with a column at each angle, and in order to get the girders above the bustle-pipe we considered using the thin-lined construction for several feet above the bosh angle, and letting the mantle at the top of this thin zone rest directly on the girders.

But on further consideration there were two features of this design which we did not like. We desired to use the steel bosh-jacket construction, cooled by external sprays, which has proved so durable and satisfactory at many plants in the South and at some in the North, and which avoids entirely the stepped bosh-wall, almost as angular and rough as a flight of steps, into which the cooling-plate construction soon wears.

When the bosh-jacket construction is used, it is necessary to realize that no matter how much brick may be laid on the jacket at the beginning of the blast, it will all soon be gone, and its place taken by a composition of carbon, slag, etc., held on the slope of the jacket by the water-cooling. This, obviously, is not a suitable structure to carry any weight, and in fact, when this construction is adopted, provision must be made for the fact demonstrated by experience that the bosh will expand downward from the mantle, where it must be supported, and the crucible upward from the hearth, and consequently space must be allowed in which this expansion can take place without injury to the structure or without shearing-off any water-pipes, and in such a position that the crack formed by the expansion will be sealed by cinder as fast as formed. This thin skerf is self-renewing and easily retained on the sloping surface of the bosh, but it is by no means so certain to be retained on the vertical section above the bosh. At the same time, the skerf on the bosh is incapable of supporting securely heavy brickwork to protect the vertical zone above it. For this reason we decided that the vertical section above the bosh should

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be brick of standard thickness, supported on the mantle in the regular way, rather than thin lined. This decision eliminated the possibility of using a high mantle supported directly on the girders.

In regard to the three-column construction, while it sets the columns back further from the furnace than with a four-column construction, and while three columns are theoretically enough to support any structure, still, in the event of a serious accident around the bottom of the furnace, it was conceivable, even though, I think, highly improbable, that one column might be sufficiently affected by heat to yield, and with the three-column construction this would throw the whole furnace to the ground.

With the four-column construction, on the other hand, if anchor-bolts to hold the structure down to the foundation are used, one column could fail completely and the two adjacent to it would carry the structure with the assistance of the counterweighting action of the column opposite to the one that failed. This, of course, would only be true if the columns were made so strong that any two would carry the structure.

Accordingly, the four-column construction was adopted and the columns designed to have an ample factor of safety with any two of them supporting the entire weight of the furnace.

In order to raise the girders above the bustle-pipe and at the same time avoid the high mantle possible only with the thin-lined zone above the top of the bosh, it was necessary to support the shell on the girders by brackets. In order to reduce the local stresses on the bottom of the shell due to these brackets, as far as possible, eight of them were used. This also had the great advantage that, by spacing the brackets so that the centers of the girders came half way between them, the load on the girders was applied relatively close to the columns and the bending moment enormously reduced. At the same time the distance from the shell to the center of bearing on the base of the brackets was increased but little over what it would have been had they come in the centers of the girders.

It is obvious that the mantle-plate would be entirely unsupported except at its outer edge and that it might deflect at the inner edge, owing to the weight of the brick-work, if some provision were not made to prevent this.

The extent to which this action could occur, in view of the

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brick being so firmly supported at the outside by the heavy angle riveted to the shell, is problematical, but it was not desired to take any chances and a cantilever-bracket was therefore designed to support the mantle-plate from below to within a short distance of its inner edge. The method by which this is done is shown by Fig. 1. The girder is shown in broken and dotted lines. The angles which rivet the bracket to the shell continue down below the mantle and support a heavy vertical plate, the inner end of which projects under the mantle-plate while the outer end is prevented from rising under the weight so applied by heavy angles fastened to it, which pass up outside the girder and take hold of the outer end of the shell-bracket. It is obvious that the angles next the shell act in tension and those outside the girder in compression, making a very rigid support for the overhanging end of the cantilever under the mantle-plate. In order to prevent any possibility of deflection of the mantle-plate between these brackets, heavy channels were riveted to the inner ends of the cantilevers in such a position that the center of the channel comes as close as desirable to the edge of the mantle-plate. This forms an octagonal ring of heavy channels, which also reinforce powerfully the inner ends of the cantilevers and make it impossible for them to fail by buckling.

The general construction is shown by Fig. 2. The section on the right-hand side is taken through the center-line of column and a tuyere and penstock are shown as if they lay in the same plane. As a matter of fact, they do not; the center-lines of the tuyeres which come nearest to the columns pass a foot or two to the side of the columns and allow ample room for rodding the tuyeres. This section was made in this way to show the minimum of room around the furnace at any point. The left-hand section is on a plane at right angles to the girder, and while the column in that view looks comparatively close to the furnace, it is, of course, in reality at the distance shown on the right-hand side.

It will be seen that four independent piers of concrete were provided and that these set back at a great distance from the foundation of the hearth of the furnace proper. These piers are at such a distance from the foundations of the hearth and so entirely separated therefrom that it seems impossible that

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the soft dirt between the two should ever transmit the expansion of the hearth sufficiently to force the piers out of position.

It will be noticed that brackets were introduced between the

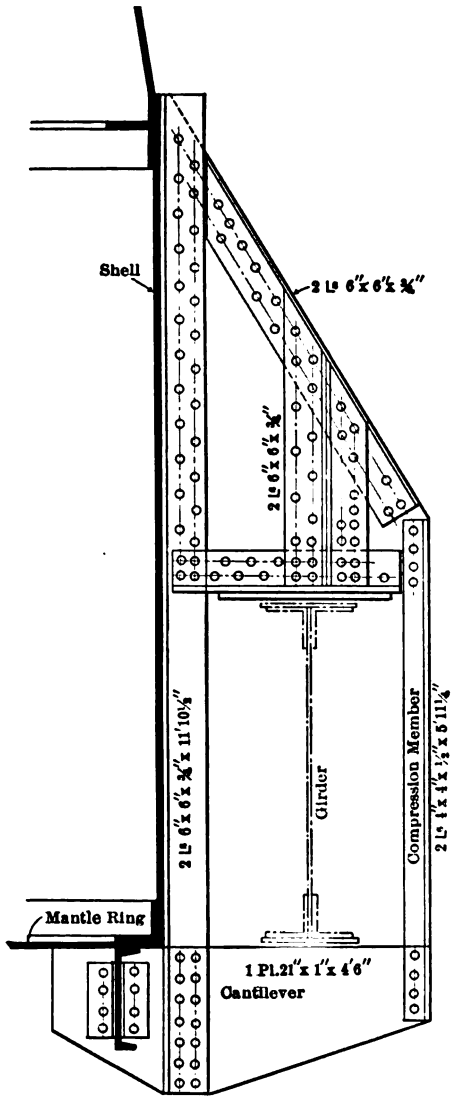

FIG. 1.—NORMAL ELEVATION SHOWING SUPPORTING BRACKET AND CANTILEVER FOR MANTLE-RING.

top of the column and the bottom of the girder on each corner to prevent any possibility of the columns failing by leaning all in the same direction circumferentially. Across the corners of

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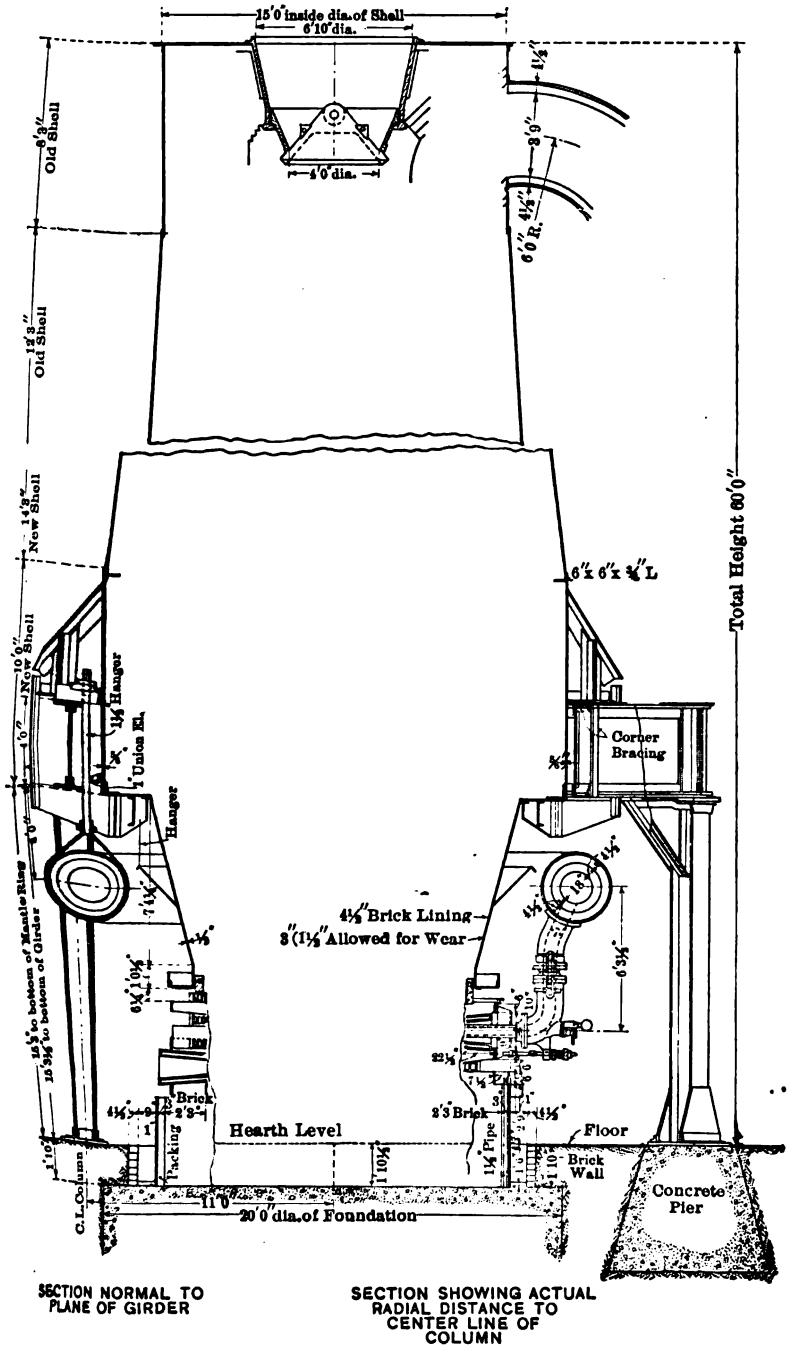

FIG. 2.—SECTION THROUGH ASHLAND FURNACE, LAKE SUPERIOR IRON & CHEMICAL CO.

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the main girders channels were riveted at top and bottom to form diagonal braces. These were latticed together vertically for greater strength, as shown in Fig. 3, which shows the structural work without the shell. The girders were so located that the inner edge of their flanges was 4 in. from the shell at the closest point, and the corner-braces were put 7 in. from the shell at the closest point, thus leaving plenty of access to the shell for anything that may be necessary, even behind the girders.

The brackets are not riveted to the girders either at top or bottom. This is for the reason that if the bottom ring of the shell should give away in any manner and carry the brackets out, then, if the latter were riveted to the girder this would put a bend in the top or compression flanges of the girders and might cause their failure. The girders and brackets not being riveted together, opportunity is provided for the bracket to slide on the girder and leave the latter undeformed under any possible condition. In addition, the bottom ring of the shell was made very strong to provide against this contingency.

The inward thrust at the top of the brackets comes on the seam uniting the bottom and second shell-plates and throws this into compression. The proportions of the brackets are such that the inward reaction from each one is relatively slight and the probabilities are that the sheets alone would be able to resist it without deformation, but in order to leave no doubt whatever on this score, a heavy 6-in. angle was run entirely around inside the shell just at the bottom edge of the second sheet in such a position as to receive the full thrust from the brackets.

This being the first time this construction had ever been used, it was not considered desirable to leave undone anything that could contribute to additional safety or provide against possibility of failure in unlikely ways. The probabilities are that some of these extra provisions for strength could be left out without detriment, but it was not considered desirable to do so in this initial installation.

In regard to the strength of the girders, they were designed to be safe with a low unit-stress, assuming that the weight of shell, top-rigging and top-house, lining, back-lining, bustle-pipe and the entire charge from the hearth to the stock-line were supported on them. This last condition is most unlikely,

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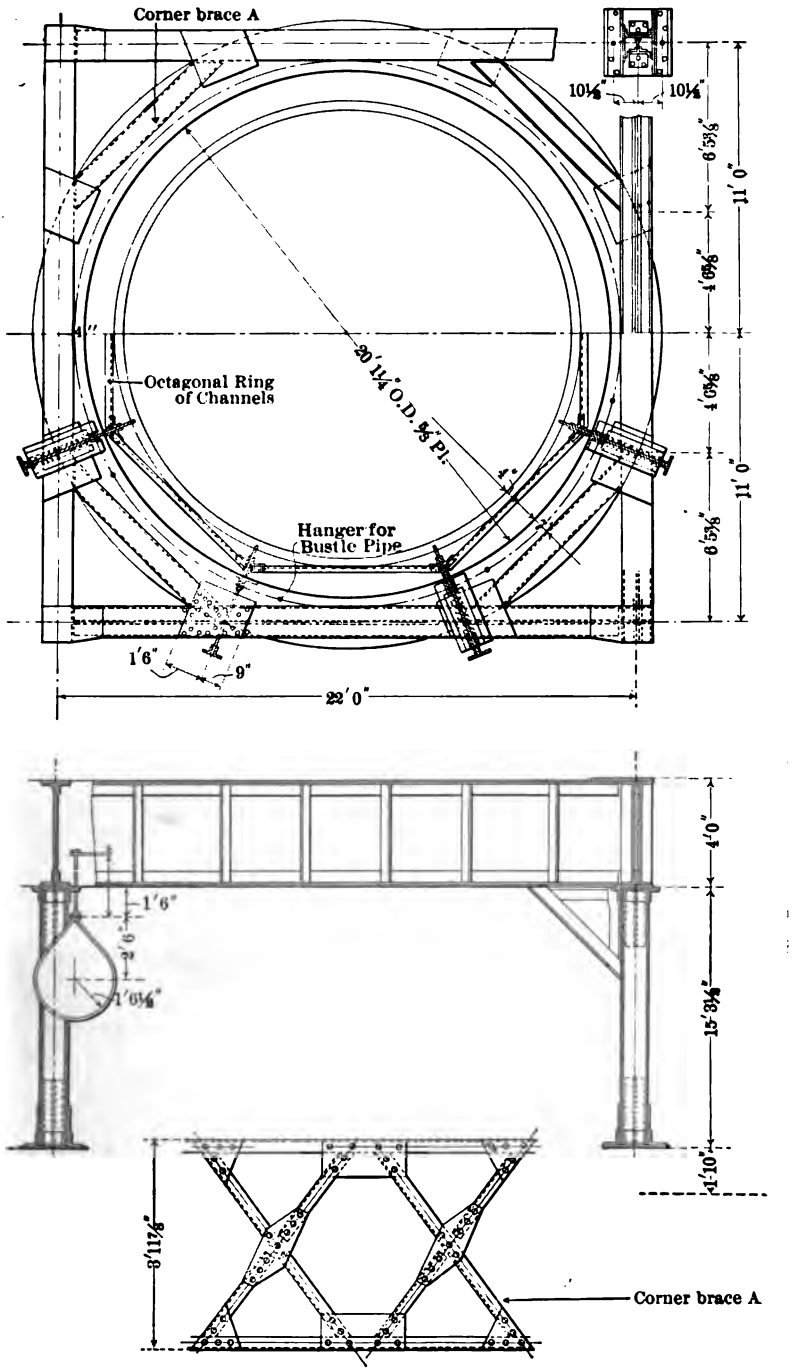

FIG. 3.—SUPPORT FOR BLAST-FURNACE, ASHLAND PLANT, LAKE SUPERIOR IRON & CHEMICAL CO.

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FIG. 4.—PHOTOGRAPHIC VIEW OF ASHLAND FURNACE OF LAKE SUPERIOR IRON & CHEMICAL CO.

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but it is conceivable that the furnace might become so badly scaffolded just above the tuyeres that the weight of the whole charge would come on to the mantle and so on to the structural work. We, therefore, provided against this contingency in the design.

The general appearance of the furnace in operation is well shown by the photograph, Fig. 4. Two men stood shoulder to shoulder as nearly in line as they could be placed between the wicket on the penstock and the nearest column, in order to show the amount of room available. In addition to this, as above stated, this tuyere is not in line radially with this column, so that the access to the tuyeres is even more unrestricted than would be indicated by the photograph. This could have been made even better than it is except that the hot-blast main was so situated as to prevent placing the columns four-square with the line of the tapping-hole, which would have been better in several respects, but we did not desire to disturb the hot-blast main and shifted the columns a foot or two to the left to miss it. It will be seen that the columns are bricked up to a height of 6 ft. to prevent any possibility of damage to them by fire. We considered also making provision to keep them filled with waste water for several feet up, but this was not considered necessary. The brick-work surrounding them is hard fire-brick laid in cement and almost indestructible.

The accessibility of the furnace for work of any kind is remarkable and I do not think that any one who had worked around a furnace of this design would ever desire to return to the old type of construction.

It will be noticed in Fig. 2 that the outside wall of the ditch is merely 4.5 in. of brick. We had some trouble with the cinder-cooler on one occasion since the furnace was blown in, and got quite a little iron into the ditch beneath it. It ran around the furnace a distance of 7 or 8 ft. This would have been sufficient to tie itself in behind one or more columns with the old construction and would have been very difficult to get out. As it was, we simply tore down the 4.5-in. wall from the outside of the ditch, pulled the "chunk" away from the hearth-jacket, out of the ditch, in one piece and replaced the wall. This accessibility is also a great advantage in working at water-

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pipes, and especially in following these from the cocks on the circle-pipe to the cooling-member that they supply.

The very short and direct connection from the bustle-pipe to the tuyeres is shown by both Figs. 2 and 4. In fact, the bustle-pipe is almost too close to the bosh-wall for greatest convenience. It is obvious that there is ample room to increase its circle and provide more room if it were ever desired to do so. Objection will probably be made to this construction on the ground of cost, but the great saving in real cost over the standard construction is one of the features which pleased us most. The first-cost of the four-column construction was less than $2,000 in excess of that of the standard construction, but of this saving about $1,000 would have been wiped out by the cost of work on the bustle-pipe had we adopted the old construction, for the following reason.

In order to have obtained the furnace-lines desired with the standard construction we would have had either to provide a new bustle-pipe complete or cut the present one down, cut it apart in six places, insert additional sections to enlarge its circle, and reline it complete, which could scarcely have been done for $1,000. The actual money cost was, therefore, a few hundred dollars greater for the four-column construction than for the standard, but we have to consider also the element of time.

Owing to the situation of the foundations for the new columns, one of these was put in before the old furnace was blown out. A second one could have been, had we known more accurately the position of the waste-water sewer, but this and the two front ones were actually put in immediately after the furnace blew out and they were done before the work of tearing down the old furnace had progressed very far. Of course, if the standard construction had been employed the old shell and columns would have had to be removed and the whole foundation exposed before the new foundation could be added. As it was, not only the foundations were built, but the structural work for the new furnace was erected by the time the bosh and the upper part of the lining of the old furnace had been torn out. We estimate a saving of from two to three weeks in time on this account alone.

When the steel bosh-jacket is used with the ordinary type of

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furnace it has necessarily to be fitted up and riveted together in several sections inside the columns, and, in fact, it is commonly attached to the edge of the mantle-ring as it is put together, with the necessity of a great deal of fitting of the vertical seams. In the present instance the bosh-jacket was built entirely independent of the furnace, and when the construction had advanced far enough to receive it, it was simply shoved in through the columns, hoisted up to place and riveted fast; the entire job taking only about two days as compared with two or three weeks that would have been required had it been built up in place in the customary way.*

We consider that we made easily a saving of five weeks on these two accounts alone. The loss of profits, overhead charges, etc., at such a plant cannot be estimated at less than $500 a day, and this saving of 35 days, therefore, represents a saving of $17,500, or, deducting $1,000 for the excess cost over the standard construction, a net saving by the use of this construction of $16,500.

It is customary to regard charcoal-furnaces as insignificant in size, but this furnace is 60 ft. high and of a size that would make from 200 to 300 tons of coke-iron per day with moderate driving. Furnaces have such an enormous capacity for exerting destructive energy that the tendency in the minds familiar with them is to overestimate their actual weight. It is doubtful if the weight of a large coke-furnace figured on the basis above

* While not germane to the subject of the present paper, it may be a matter of interest to those who use or contemplate using the steel bosh-jacket construction that this jacket has not a single rivet in it except those at the top which fasten it to the mantle-ring, and those in the bottom which hold the trough. The vertical seams are welded up solid with the oxyacetylene blow-pipe. This was done principally for the reason that past experience had shown the necessity of having the surface of these jackets as smooth as possible in order to have the water follow them from top to bottom. The least obstruction will throw the water off and allow a dry spot to form below it, which causes burning or cracking. In order to get satisfactory results with a riveted jacket it is necessary for all rivets to be countersunk, and even then the smoothness of the seam leaves much to be desired, and its strength is less than with headed rivets. It is also much more difficult to prevent burning at the seams on account of the double thickness of metal at that point. By welding up the jacket in one solid piece and chipping and grinding the seam after the welding was done, we got a much smoother job than was possible with riveting, and one which the water follows as perfectly as it does the rest of the jacket. At the same time the objection to the double thickness of metal at the joints is overcome. I believe that this is undoubtedly the correct way to make these jackets.

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described would exceed 2,000 tons, and there is no difficulty whatever in carrying a load of this amount with a construction of this kind.

It may be well to point out that this construction is particularly adaptable to the thin-lined type of furnace because the weight of the lining in such a furnace is an insignificant fraction of that of a standard thick lining, and because some outside structure is generally necessary for supporting the platforms, spray-apparatus, etc., even if not the furnace-top proper. This can all be done with great advantage from the girder-frame if the type of construction here described be used.

It is well to point out also that if the cooling-plate construction of the hearth be used, so as to form an adequate support for a thin lining in the zone immediately above the bosh, then the high mantle, which results from that construction, may be directly supported on the girders without the use of brackets. Methods can also be devised without difficulty for supporting the lining of the thin zone with the steel bosh-jacket construction by providing a narrow water-cooled false mantle at the bosh angle, but we did not consider this construction necessary for our conditions.

Conclusions.

1. A furnace designed on the plan described is not only feasible but a safe and substantial construction.

2. This construction is especially well adapted to the thin-lined or partly thin-lined furnace.

3. Its convenience in operation is so much greater than that of the older type of furnace that there is no comparison from an operating point of view.

4. The time to be saved by the use of this construction, at least in rebuilding, is so great as to constitute a money-saving of far greater extent than the slight excess cost of this construction over the standard.

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Blowing-In a Blast-Furnace.

BY R. H. SWEETSER, COLUMBUS, OHIO.

(Cleveland Meeting, October, 1912.)

THERE are probably as many variations of the method for blowing-in blast-furnaces as there are furnace superintendents. That some of these variations are poor practice is shown by the troublesome and sometimes disastrous experiences of many furnace-men, even within the last few years. It is the object of this paper to put on record a blowing-in practice that has gradually developed to the point where the results are sure, safe, and satisfactory; and also to bring into discussion several variations that are questionable.

The blowing-in of a modern blast-furnace is one of the greatest of metallurgical operations. On its degree of success depends much of the future life of the lining and the subsequent behavior of the furnace; and it involves possibilities of danger to the men engaged in the work. The objects to be attained are the heating of the interior of the furnace to the high temperature necessary for smelting iron; the starting of the smelting process; and the gradual movement of the stock in its downward course. The most critical points are the placing of the kindling-wood, the "bringing down" of the gas, the closing of the iron-notch, and the first cast. The lighting of the first gas, the opening of the cinder-notch for the first flush, and the running of the first cast are anxious times for the superintendent and the furnace-men.

Drying-Out the Furnace.

As soon as a furnace-lining is completed it should be thoroughly dried out, whether it is to be blown-in soon or not; and the longer it is dried, the better for the furnace. At least two weeks is desirable; but good results can be had with 10 days of firing. The crudest and least desirable method of drying is to build a wood fire in the bottom of the furnace, throwing the wood in through a tuyere-arch, and using the iron-notch for

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draft. The accumulation of wood-ashes on the hearth-bottom keeps the brick from heating thoroughly, and considerable time is lost in removing the ashes.

An effective and easily-controlled method is to build a "Dutch oven" outside one of the tuyeres and use coke for fuel. This method may be intermittent or continuous, a desirable feature, since sometimes it is necessary to pack plates or coolers on the day-turn, and dry-out on night-turn. Coal could be used for fuel, but coke is cleaner and gives a hotter fire.

The "Dutch oven" can be built at the iron-notch, but often it is necessary to fit up the iron-notch gun and the gun-bar; and then the oven would be in the way. With the oven at a tuyere-arch, the hearth-bottom can be well heated by a wood fire built right in the casting-trough; the strong draft will draw the heat through the iron-notch. Fig. 1 shows a simple oven for a tuyere-arch; if used at the iron-notch, the neck of the flue would have to be turned downward.

Pipe for the Iron-Notch.

In order to burn completely all the kindling-wood below the tuyeres before the slag falls into it, and also in order to heat up the hearth-bottom thoroughly, it has long been customary to place an iron pipe, 3 or 4 in. in diameter and about 10 ft. long, in the iron-notch. This pipe should project from 6 to 18 in. inside the furnace-wall. If the hearth-bottom is much below the level of the iron-notch there is no need for the pipe to project much beyond the inside of the walls, but if the hearth-level is almost up to the bottom of the iron-notch, the pipe should project farther. In either case the part of the pipe inside the furnace should be thoroughly covered over with stiff clay, so as to protect it from any molten slag or iron.

The pipe should be in the center of the iron-notch, and as nearly level as possible. Near the outer end of pipe there are four holes, 0.75 in. in diameter, in which to insert hooks for pulling out the pipe when the time comes. As soon as the blast is on the furnace, the gas will come through the pipe out into the iron-trough, where there should be a wood fire to ignite it immediately. As the volume of gas increases, the gas-flame at the end of the pipe gets hotter and roars more, but does no harm. It is my practice to let this gas burn at the iron-notch

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for 4 hr. after starting the blast, and then to pull the pipe and shut the notch.

Shutting the iron-notch is one of the critical stages of blowing-in; but if everything is in readiness the work can be done in

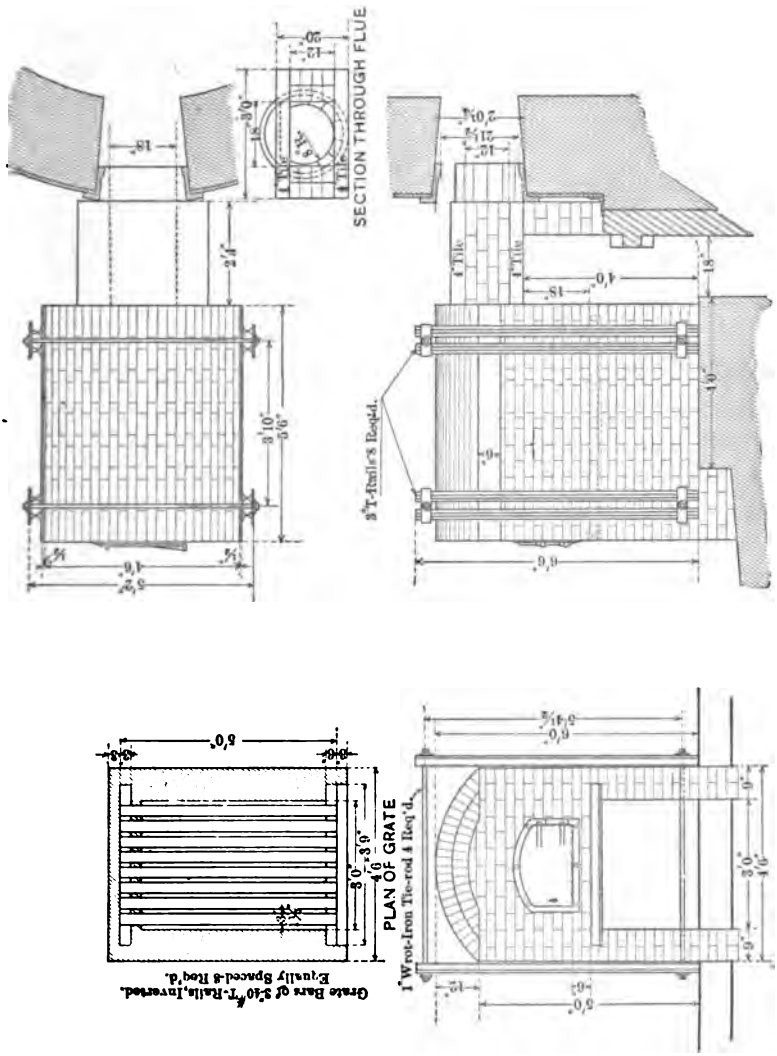

FIG. 1.—"DUTCH OVEN" FOR DRYING-OUT BLAST-FURNACES.

less than 3 min., with an iron-notch gun and a steam-cylinder attachment on the gun-bar. The iron pipe should be turned in the iron-notch so as to loosen the clay; this can be done by inserting a bar or hook in one of the four holes near the outer

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end of the pipe. A quick jerk on the hook will pull the pipe clear of the iron-notch, and then the gun is swung around quickly and forced into place with the steam-cylinder.

Filling the Wood.

The placing of the kindling-wood in the hearth and bosh is such an important part of blowing-in a blast-furnace, that the superintendent or his assistant usually gives personal attention to each detail, staying inside the furnace to see that every stick of wood is properly placed. The simplest method (as is often the case in furnace-practice) of putting wood in the hearth has proved to be the best in every way. David Baker taught it to me 20 years ago, and he doubtless learned it from some old furnace-man at Steelton. No fancy scaffold is made, but cord-sticks are packed as tightly as possible, in alternate layers at right angles to each other, from the hearth-bottom up to the level of the bottom side of the tuyeres. The wood is all passed into the furnace through one tuyere-arch with the cooler left out.

At the Algoma Steel Co., I first used the "Algoma cones" at the tuyeres. These are made by placing dry building-laths in front of each tuyere with the inside ends coming to a point towards the center of the furnace, and the other ends placed all around the nose of the tuyere. The spaces between cones are filled up with short sticks, split cord-sticks and any dry kindling available, and the whole surface is leveled with the exception of a place about 3.5 by 4 ft. in front of the tuyere where the wood is passed in. A vertical shaft about 10 ft. high is here made by nailing four posts (4- by 4-in. sticks or small saplings) in position. Two posts will be close to the wall on either side of the tuyere-arch and the other two will stand about 4 ft. from the wall. Cord-sticks are then piled on end all across the furnace and around the shaft; then a second tier of cord-sticks is placed in as tightly as possible. Usually the top of this second tier will be below the top of the bosh, and cord-sticks are laid against the walls all around the furnace to protect the brick and plates from falling stock.

When this is done enough wood is passed inside the furnace and up the shaft so that when the last man "backs out" he can fill the shaft solid with wood. All the men except two go outside, and the tuyere-cooler is set in place. The two men pack

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the cooler from inside with clay, and place the kindlings in front of the tuyere, then "back out" up the shaft. The wood is then thrown down into the shaft and packed in solid, and the two men are hoisted out at the top of the furnace in a boatswain's chair.

Filling with wood in the above manner accomplishes perfectly the three things desired, namely: the heating-up of the hearth without leaving any appreciable amount of ash; the ignition of the coke-charge and the starting of the smelting process; and the movement of the stock with such gentleness that the settling on top, within 2 hr. after the blast goes on, cannot be detected except by using the gauge-rod.

Coke on the hearth-bottom and built-up scaffolds do not usually attain all these points, and do not produce good iron on the first cast.

Filling the Stock.

On top of the wood, regular charges of coke with just enough limestone in each charge to flux the ash of the coke, are dumped in the usual manner. The size of this coke "blank" is more often too large than too small. I am indebted to Frank P. Howe, of Philadelphia, for learning the advantages of using about 20 tons of coke instead of 40 or more. The next set of charges has an equal amount of coke plus cupola-slag, and enough limestone to flux the coke and slag. Then follow the ore-charges with gradually decreasing amounts of slag. Charcoal-furnace slag is excellent for the slag-charges, but is usually harder to get than clean cupola-slag. If the slag is not granulated it should be broken up to about 2-in. size. A good volume of easily-melted slag is needed to fill up the hearth and receive the first iron that comes down.

A charging schedule that has worked successfully several times is as follows:

Charges.	Coke. Pounds.	Limestone. Pounds.	Cupola-Slag. Pounds.	Ore. Pounds.
6	6,510	730	0	0
6	6,510	4,000	4,000	0
5	6,510	4,300	4,000	1,800
5	6,510	3,600	3,000	2,400
5	6,510	3,000	2,000	3,600
5	6,510	3,300	2,000	5,000

These materials filled a 75- by 18-ft. furnace.

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The first five charges after the blast went on had 1,000 lb. of slag and 6,400 lb. of ore; then the slag was omitted and the ore gradually increased to full burden.

When the stock gets within about 20 ft. of the top, it is a good plan to take measurements of the surface of the stock, and to make careful observations of the distribution; at this time, gauge-rods should be tested as to accuracy in readings.

Putting the Blast On.

I believe in putting the blast on at once, and using hot blast to light the furnace whenever stoves previously heated with gas are available. If the volume of blast is increased properly, the furnace will not "hesitate," but will move off easily, and the first slip will be indefinitely postponed. If the stoves have not been previously heated with gas, then red-hot rods are thrust in through the tuyeres to light the kindlings.

If the furnace is a single stack and the gas-flues are cold and empty, the method of "bringing down the gas" is different from the way when there are one or more furnaces in blast and delivering gas into the same gas-mains.

When the blast is turned on a single stack, both bells and the top bleeders are left open for a while and not closed until there is a steady and voluminous flow of gas at the top of the furnace. The big bell, and then the little bell, are closed, and all the gas goes out at the bleeders. All gas-burners on stoves and boilers are tightly closed, and all fire is kept away from dust-catchers and cleaning-doors. The gas-mains are gradually filled with the dense white furnace-gas by partly closing the bleeders. At the farthest end of the gas-main, gas is allowed to escape through a valve or a cleaning-door, so as to expel all the air in the mains. When gas is leaking through all the cracks of doors and burners, and there is pressure inside the gas-main, the last gas-burner under the farthest boiler is slowly opened, and the gas is lighted by the fire on the boiler-grate. The bleeders are then closed so as to force all the gas into the mains. By having the mains full of gas at a pressure of a few inches of water, the danger of explosions is averted. The very dangerous quality of the blowing-in gas is shown by the analyses I made and recorded in our *Transactions*,[1] but all trouble can

[1] In my paper, Analysis of Blast-Furnace Gas While Blowing-In, *Trans.*, xxviii., 608 to 613 (1898).

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be avoided by proper handling of the gas. Great care must be taken not to open gas-burners faster than the volume of gas will allow; as the volume and pressure increase, the burners at the stoves can be opened.

In the cases of "twin" furnaces, all these precautions can be omitted and the gas from the new furnace can be soon turned into the gas-mains by closing the bells and bleeders. Of course, the gas-valve between the new furnace and those already in blast must be opened just before the blast is turned on.

I will not attempt to give in exact cubic feet the amount of blast that should be blown during the first hours of blowing-in; but the volume at all times during the first day or two should be sufficient to prevent much blast-pressure on account of "hesitating" in the settling of stock. While the wood is burning the blast-volume should be sufficient to prevent gas from coming back in the tuyeres by reason of the pulsations of the engines. I usually start with 25 per cent. of the normal blast.

The First Flush and First Cast.

The first flush is anxiously looked for in 10 or 12 hr. after the blast goes on. Where slag is used in filling the furnace, a considerable amount should be flushed out before any attempt is made to open the iron-notch.

In opening the iron-notch for the first time there should be no trouble whatever, providing the keeper drills in on the level and through the soft clay that fills the hole left by the 4-in. pipe. Sometimes a big tapping-bar is driven into the soft clay as soon as the iron-notch is shut for the first time, as described above. But I have found that there are several disadvantages in this and now prefer to leave the bar out.

Some Records with the "Columbus Method."

Date.	July 15, 1908.	Aug. 17, 1909.	May 21, 1912.	Aug. 20, 1912.
Size of furnace, feet.....	75 by 18 Single.	75 by 17 "Twin."	75 by 18 Single.	75 by 18 "Twin."
Blast on.....	8.03 a.m.	8.25 a.m.	4.03 a.m.	3.42 p.m.
How lighted.....	Hot rods.	Hot blast.	Hot blast.	Hot blast.
Stock settled.....	9.45 a.m.	9.20 a.m.	6.10 a.m.	5.20 p.m.
Iron-notch shut.....	.....	11.05	8.00	8.22
	12.35 p.m.	11.11 a.m.	8.05 a.m.	8.25 p.m.
First flush.....	6.45 p.m.	6.37 p.m.	4.20 p.m.	8.00 p.m.
First cast.....	10.10 p.m.	9.25 p.m.	7.30 p.m.	12.30 p.m.
Silicon, per cent.....	4.33	2.73	3.53	1.79
Sulphur, per cent.....	0.044	0.028	0.017	0.035
Tons pig first week.....	1,302	1.137	1,410	1,263

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The furnace blown-in Aug. 17, 1909, had very hot stoves to start with, and the blast-temperature exceeded 1,400°. On this account the blast-volume was increased more rapidly than usual, and the first cast (34,620 lb.) was only 13 hr. after the blast went on.

The furnace blown-in Aug. 20, 1912, had no bottom-brick on top of the salamander that had been left in, and on account of the increased depth of hearth the cinder was longer than usual in filling up to the cinder-notch.

[SECRETARY'S NOTE.—For earlier contributions to the Institute covering this subject in whole or in part, see the following papers in the *Transactions*.—J. S.]

Vol.	Page.	Title.	Authors.
VIII.	348 to 354	Blast-Furnace Working.	Julian Kennedy.
XVII.	138 to 140	Large Furnaces on Alabama Material.	F. W. Gordon.
XVIII.	379 to 391	Filling and Blowing-In at the Durham Blast-Furnace.	B. F. Fackenthal, Jr.
XVIII.	427 to 436	Peculiar Working of a Blast-Furnace.	N. B. Wittman.
XX.	262 to 267	American Blast-Furnace Practice (Discussion).	James Gayley, J. E. Johnson, Hugh Kennedy, John Birkinbine, E. C. Pechin, Julian Kennedy.
XXIII.	370	Review of American Blast-Furnace Practice.	E. C. Potter.
XXVIII.	608 to 613	Analysis of Blast-Furnace Gas While Blowing-In.	Ralph H. Sweetser.

DISCUSSION.

J. E. JOHNSON, JR., Ashland, Wis.:—There is one method of bringing down the gas which Mr. Sweetser has not mentioned, and which is very useful. John W. Dougherty, now of the Crucible Steel Co. of America, is entitled to the credit for the common knowledge of a plan. This consists in turning steam into the downcomer or dust-catcher before lighting the furnace, so as to form a steam cushion between the air and the gas. The danger of course comes, not from the air alone or from the gas alone, but from the mixture of the two, which is produced when the gas first begins to come through. The steam cushion between the air and the gas prevents the possibility of an explosive mixture being formed. Mr. Dougherty has a patent on this process, but I believe it is established that the process was in use elsewhere some two years before his patent was applied for, and therefore the patent is of doubtful validity,

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which is unfortunate, because Mr. Dougherty undoubtedly evolved the idea independently of its previous use.

In blowing-in a furnace, I think it good practice to speed the engine up for a few minutes to a considerably higher speed than you expect to use after the gas is brought down. There is at first a very slow action, due to the great quantity of voids in the furnace to be filled up, and the progress of the gas is exceedingly slow, if the slow blowing which is suitable for blowing-in be used from the very beginning. By running the engine at 50 or even 100 per cent. greater than the blowing-in speed for a few minutes, the gas is brought through promptly, and the dangerous period very greatly shortened. After the gas is brought down and safely lighted, the engine can be put down to its proper speed. This need not be more than 10 or 15 min. I lit a furnace about a month ago, following this practice, and the gas was down and lighted at all burners in about 10 min. from the time the wind was turned on, without a sign of a puff of any kind. We used the steam cushion, and also led the gas to the far end of the flue, opening the boiler- and stove-burners after the thick white first gas had pushed the steam out of the opening.

In connection with the stopping of the tap-hole, a very good plan is to build a considerable pile of clay, mixed with coke- or coal-braize, over the tap-hole, and extending back well into the hearth. The pipe described by Mr. Sweetser is arranged to project through this into the furnace. When the tap-hole comes to be opened, it is easy to drill through this soft mass to the skull which the heat of the hearth bakes on its surface, and a bar can then easily be driven through this skull.

This plan may save a great deal of trouble when it comes to opening the iron-notch. The principle is the same as that of the cinder-notch which projects into the molten matter in the hearth. It is also the same as the deep hole kept by the aid of the steam-gun after the furnace is in operation.

Mr. Sweetser's plan for pulling the pipe out of the tap-hole when it is desired to close the hole, is quite the best that I have heard of.

J. J. Howard, Dover, N. J. (communication to the Secretary*):—No doubt there are as many different methods of start-

* Received Jan. 25, 1913.

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ing a blast-furnace as there are men who assume that responsibility. Having blown-in a furnace a few years ago in a manner similar to that outlined by Mr. Sweetser, things went along so well at the start that I had time for a little reflection on what was going on inside the furnace. I saw the wood turned to charcoal practically, and it occurred to me, why should I use wood when I could put in the charcoal in the first place, and thus save a lot of time, and also eliminate the dangerous gas from the wood, which is the cause of most of the accidents in starting up a furnace. I had a furnace out for repairs and decided to use no cord-wood in blowing in. I put in about a carload of charcoal and used no wood except some shavings and oily waste at each of the tuyeres. The furnace had 16 tuyeres and was 100 by 22 by 14.5 ft. Hot rods were put in each tuyere and the blast started immediately at 11 a.m. Within 5 min. the air was all driven out of the stack and the gas lighted of its own accord at the top of the furnace. It was allowed to burn for 5 min. longer and then the bell was closed and the gas taken to the last stove and the last boiler on the gas-main. It was all done so quietly that it was hard to realize that the gas was down. The tuyeres were as bright as if the furnace had been in blast for a long time. I have learned since that others have used the same method, but they had not passed the good advice along.

With regard to the blank charge of coke in starting, I think this amount is governed by the size of the furnace and the condition of the ore, whether soft or refractory. I find that 0.5 lb. of ore to 1 lb. of coke on top of the blank is light enough—the ore is increased every five charges until the furnace is full, a ratio of one of ore to one of coke or a little better, if the hearth is not new.

I have found it best to use lean ores and to have a large volume of slag in starting, so as to avoid making any high-sulphur iron, and also to have the manganese content in the iron as near 1 per cent. as the ore mixture will allow.

Hot stoves to start with are an advantage, but many a trouble, after the first day in starting up, may be traced to carrying high heat on the blast during the first few days.

Blowing-in a furnace was formerly attended with a great deal of ceremony, but to go to the other extreme, I know of a

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manager who says it is no more difficult to blow-in a furnace than to start a fire in a house heater.

F. FIRMSTONE, Easton, Pa. (communication to the Secretary *):—Neither in Mr. Sweetser's paper nor in those in our *Transactions* referred to in the Secretary's note at the end of it, is there any special mention of the advantages resulting from the free use of properly selected blast-furnace cinder in filling a furnace for blowing-in. The following notes on the blowing-in of a coke-furnace, 75 ft. high, 18 ft. in diameter, may be of interest.

The furnace had been very well dried before filling was begun. A flue about 9 in. square in the clear, formed of three planks, was laid on the bottom from the tapping-hole to the center of the hearth, and enough light dry wood heaped around the end of it to insure the ignition of the cord-wood next put in. The filling then proceeded as follows:

Two tiers of 4-ft. cord-wood placed on end as in filling a coal-pit (meiler), the first tier resting directly on the bottom of the furnace, then—

1st. 36 barrows (5 cwt. each) of coke, 9 tons.
2d. 36 barrows (5 cwt. each) of gray furnace-cinder, 9 tons.
3d. 6 rounds, each of
12 barrows (5 cwt. each) of coke.
9 barrows (5 cwt. each) of cinder.
4 barrows (5 cwt. each) of ore.
2 barrows (5 cwt. each) of limestone.
4th. 6 rounds, each of
12 barrows (weight as above) of coke.
6 barrows of cinder.
6 barrows of ore.
3 barrows of limestone.
5th. 10 rounds, each of
12 barrows of coke.
3 barrows of cinder.
8 barrows of ore.
4 barrows of limestone.
6th. 6 rounds, each of
12 barrows of coke.
2 barrows of cinder.
10 barrows of ore.
5 barrows of limestone, which about filled the furnace.

After the blast was on we filled per round, 8 barrows (5 cwt. each) of coke, 8 barrows (5 cwt. each) of ore, and 4 bar-

* Received Jan. 25, 1913.

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rows (5½ cwt. each) of limestone, which was suitably increased as the working of the furnace indicated. The bell not being in place, the barrows were dumped on the plates at six places equally divided around the top and shoveled into the furnace. When the furnace was nearly full it was found that the surface of the material was practically level, so that the shoveling had been properly performed. This was of great importance in filling anthracite-furnaces, but with coke it seems to be sufficient to dump into the hopper and lower the bell as in regular work. Dumping the barrows directly into the open top is sure to result in heaping the stuff in the middle o the furnace and thereby causing the larger pieces to roll towards the walls.

The furnace was fired through the flue at the tapping-hole at 6.25 a.m. June 2, 1902; in 10 min. smoke showed at the top; at 7 a.m. we lighted at the tuyeres by putting a little burning charcoal in each and blowing lightly with cold blast; at 8.07 a.m. we began to blow through all the tuyeres (having previously closed the tapping-hole), using No. 3 stove; at 8.11 a.m. closed the bell and after some few minutes waiting, admitted the gas to the boilers; at 8.45 a.m. to No. 1 stove and at 9.45 a.m. to No. 2 stove; at 10.05 No. 3 stove off air, No. 1 on, temperature of blast 450° F; at 10.10 a.m. began to fill. At 11.10 a.m. the cinder showed at the tuyeres and the first flush was tapped out. It was hot and lively but rather dark-colored. The second flush was tapped at 12.05, like the first; the fourth, at 3 p.m., still dark but hot; the fifth, at 4.50 p.m., hotter and beginning to turn gray. The first cast, on the morning of June 3, was white, the second, in the a ternoon, gray forge, and subsequent casts of No. 1. There was no trouble with the tapping-hole.

We blew about 6,000 cu. ft. of air per minute for the first 24 hr.

It had been planned to light the furnace in the evening, and let it draw until the wood in the hearth was completely charred, and then blow early in the morning. So much time was lost in getting the bell into place and getting it to work properly that this plan was abandoned and we blew at once after lighting in the morning. On other occasions, by lighting about 6 p.m. at the flue at the tapping-hole, the wood was completely charred and burning charcoal at the tuyeres by mid-

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night. Flue and tuyeres were then closed tight and not opened until the pipes were put up to blow early the next morning. When thus managed a furnace always gave gray cinder and gray iron from the start. In either case the first flush of cinder should be had in from 2.5 to 3.5 hr. after blowing.

With anthracite there is not much danger of premature smelting or sintering of the bed of cinder while the wood is charring, but with coke-furnaces, which commonly have a stronger draft, I am told that this has caused trouble. In any case it is best to blow too soon rather than too late, the chief difference seeming to be that by allowing the furnace to draw, the first of the ore is so much heated by the time that the blast goes on that it is all reduced and the first iron and cinder are gray, otherwise enough of the ore passes into the cinder to darken it and cause the first iron to be white.

The old furnace-men understood perfectly the importance of producing a "good" cinder and plenty of it, from the start, but this was a matter of some difficulty even when the furnaces were "scaffolded down." If too little burden were put on, the first cinder came into the hearth in driblets and was likely to set there instead of coming in sufficient volume to establish a good flow over the dam; if too much, there was danger of a cold cinder highly charged with iron, very destructive to the hearth (the "scouring" cinder referred to in old metallurgical books), accompanied by cold white iron, pretty certain to give trouble at the tapping-hole; moreover, in those days there was practically no help from analysis so that there were many chances against getting a "good" cinder at the start.

I have never heard who had the happy idea, that by putting in plenty of cinder of the kind you desired to have, you would get from the start an abundant flow of nearly the same kind of cinder (modified only by the ashes of the fuel and by any ore which it would dissolve) and thus greatly reduce the chances of trouble until the furnace came into normal work. The plan came into use at some of the furnaces in the Lehigh valley between 1845 and 1850. A very minute description of it was prepared by the late Dr. Thomas Egleston, and sent by him to M. Ed. Sauvage with the request that he should translate it for the *Annales des Mines*. This was done, and the translation may be found in *Annales des Mines*, 8th ser., tome ix., p. 313 (1886).

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As there stated, the cinder used should be gray and basic, for the percentage of silica in it will be notably increased by the ash of the first fuel burned.

I have never seen the least trouble arising from unburned wood in the hearth or with the tapping-hole. In fact, with a large volume of hot cinder filling the hearth, it is so thoroughly heated up that any unburned wood must be speedily charred, and the iron, when it comes, falls into a bath of cinder just as it does when the furnace has come to normal working.

There is no danger from explosion if the dampers on the gas-mains be really tight, as they should be. A rough calculation of the cubic content of the gas-pipes may be made, and the gas allowed to escape through explosion-doors left open for the purpose, and the dampers to the stoves and boilers opened cautiously, one by one, only after the engines have delivered a volume of blast eight or ten times the cubic content of the mains.

I have seen 30 or 40 blow-ins, as above described, and only on two occasions did anything go wrong. In 1877 we blew in No. 5 Furnace at Glendon, which was filled with a mixture, half of brown ore and half of magnetic ore from the Hurd mines, Hurd-town, Morris county, N. J. This latter was a very rich magnetic ore much like the Cheever ore from Port Henry, Lake Champlain, *i.e.*, it broke up readily into pieces from say half an inch in diameter to the size of shot. The pieces showed bright polished faces like crystals, although no distinct crystal forms could be discovered. It often averaged more than 66 per cent. of iron. The mixture had been used for some two years for the production of foundry-iron, in place of our regular mixture, without any difficulty, but we had never filled it for blowing-in. By that time, blowing-in had become a matter of routine, and was left largely to the founder. For some reason he greatly reduced the weight of cinder charged on the first coal without consulting any one higher up. The furnace started as usual, blast on at 11.40 a.m., first cinder 3 hr. later and was normal, but successive flushes, instead of improving, grew worse and worse until by 8 p.m. the cinder would not leave the tuyeres, and filled all the blow-pipes. By putting a tuyere in the tympe we recovered all the tuyeres, and had the furnace working normally in about four days, and no apparent bad after-

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effects were noticed. After the trouble was over it was attributed to the relatively small weight of the first cinder filled, 6 tons instead of say 10, but in 1879 a second attempt to blow-in on the same ore-mixture when the weight of cinder was normal (10 tons) resulted pretty much the same as the trial in 1877, and we were again forced to blow in the tympe before we got the furnace going. The trouble was no doubt in the physical properties of the Hurd ore. We shoveled the materials into the furnace in both cases, and as it was 72 ft. high there was much breakage of the Hurd ore, and probably a good deal of it sifted into the interstices of the bed of cinder, and running into the hearth with it, charged it with iron oxide. It is also very likely, but not proved by experiment, that such rich ore with the individual grains having a very smooth polished surface was not easily acted on by the gas but came down as a mixture of ore [2] and spongy iron, plenty of which was, in fact, found at the tuyeres. We never again tried to blow-in on this mixture, but were I forced to use a similar one I should still further increase the weight of the first charge of cinder, and by lowering the ore in buckets and carefully spreading it, reduce the chances of fine ore sifting through before it could be properly reduced.

R. H. Lee, Lebanon, Pa. (communication to the Secretary *):—In my experience I have found the principal points to be observed in blowing-in a blast-furnace are the increases in the volume, and the raising of the temperature of the blast. A failure to gauge these properly is probably the original cause of all troubles experienced in blowing-in. The placing of the kindling and cord-wood is to my mind not very important, in fact, almost a matter of indifference, because it is there simply to light the coke. Lighting a blast-furnace differs only in magnitude from lighting a stove, and if enough kindling and cord-wood are used the coke is bound to become ignited, and if judgment is used in handling the blowing-engines, the coke will be lit uniformly all over the bosh. When using natural draft this point takes care of itself.

I see that Mr. Sweetser has broken away from the custom that obtained for many years of putting a scaffold in the hearth, and filling the bottom of the hearth to the cinder-notch with

[2] Bell, *Chemical Phenomena of Iron Smelting*, p. 58 and following (1872).

* Received Jan. 20, 1913.

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coke. Personally, I could never see any difference in the go-off between a furnace with the customary scaffold in the hearth and one without, and for a number of years I have dispensed with it altogether, as it simply makes a delay of some hours while it is being constructed before the wood can be put in.

Putting coke in the bottom of the hearth ought theoretically to give a hot hearth, as the blast going down to the bottom in order to escape through the iron pipe placed through the iron-notch must ignite the coke and raise it to incandescence, but as in many other cases theory does not always work out right, and I have known of several cases where a layer of iron has chilled upon the top of the coke in the hearth, preventing anything from reaching the iron-notch. In one case the furnace had an ugly explosion by the burning of the cinder-notch and monkey, because the iron could not get below the cinder-notch on account of a layer of solid iron on top of the coke. A hearth can be got just as hot by filling the hearth with wood, and using the iron pipe in the iron-notch. Should the pipe, however, be omitted, as is sometimes done, there is liability to find the hearth cold when it is time to tap for iron.

Whether a furnace is started by natural draft, or by the blowing-engines, does not seem to make a difference in the success of the blow-in. In the former case the time until the furnace is up to its burden and output is a little longer, say from twelve to sixteen hours, but not quite so large a coke blank is required. The important point to observe when starting with the blast on is not to go so fast that the ore is brought down insufficiently heated, and reduced. I have always advocated using as little wood as possible. Enough to make a good hot fire and insure the ignition of the coke is all that is necessary. As Mr. Sweetser says, the simplest way is generally the best way, and I find that cord-wood up to about one foot below the tuyeres, covered with about three feet of kindling, a ball of cotton waste saturated with coal oil being placed about six inches from the nozzles of the tuyeres, and upon this layer of kindling a rank of cord-wood, is amply sufficient for any furnace. The wood below the tuyeres I have placed on end, laid horizontally, and thrown in helter skelter, and have never been able to note that either of these ways had any advantage over the others, unless that throwing the wood in at random saved a little time. The pipe

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in the iron-notch allows the gases to escape, and the blast to penetrate the hearth, and insures that the wood will be burned and the hearth hot. With a small quantity of wood there is no drop of the stock, and there is less of the explosive wood gases made, the dangerous character of which Mr. Sweetser showed in our *Transactions*.[3].

Another point, I think of great importance, which I notice Mr. Sweetser observes, although he does not call attention to it, is the use of a refractory slag at the start. The refractory slag insures a hot hearth at once. There is always an excess of fuel present when blowing in, so that there is no danger of not having a hot and fluid slag if ordinary judgment and knowledge is used; and instead of having a few casts of high-sulphur iron, good iron is obtained at once, while the hearth cleans itself quickly, and in a day or two the furnace will be on regular casting-hours regardless of how much iron the hearth must hold. I remember when I first started in the blast-furnace business that a fad for blowing-in cold swept over the country, the theory being that by putting in but little coke the melting-point from the start was located about where it would be in normal working, and although the hearth would be cold at first, it would gradually become heated, and there would be no disturbance of the melting-point or scaffolding. By this method of blowing-in the hearth was certainly cold, but in regard to absence of trouble it did not work out so well, and the amount of high-sulphur iron made was enormous. One or two such blow-ins were generally sufficient and the furnace superintendent turned his attention to means for getting the hearth as hot as possible as quickly as possible.

R. H. Sweetser, Columbus, Ohio (communication to the Secretary *):—The use of charcoal instead of wood, mentioned by Mr. Howard, has three objectionable features: first, the liability to ignite before the desired time; second, greater difficulty in handling; and third, the extra cost. Moreover, if "the gas lighted of its own accord at the top of the furnace," it surely was more explosive and dangerous than the dense smoky gas that comes from the wood.

Sometimes wood is filled into a furnace "as soon as a man

[3] *Trans.* xxviii., 608 to 613 (1898).

* Received Apr. 29, 1913.

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can hold his hand on the brickwork in the hearth" after drying out. Charcoal is likely to ignite under such conditions; and it actually did catch fire at "the Soo" when blowing in the big charcoal furnace in 1905. At that time we handled the charcoal in sacks, which were passed through a tuyere arch into the hearth until the charcoal was level with the tuyeres. The balance of the charcoal was dumped through the bells in the regular way of skip filling.

In most places charcoal is expensive and hard to get; any dry, firm wood can be used for the kindling, and at most plants can be had for the cost of handling.

Mr. Lee speaks as though it were unnecessary to use care in placing the wood in the hearth. But if the sticks are regularly and solidly packed into the hearth the coke will come down to all the tuyeres at about the same time, and there will be no danger of the furnace working faster on one side while blowing in.

I am glad that Mr. Lee gave instances of the bad features of using coke instead of wood in the hearth.

Mr. Firmstone's description of the use of blast-furnace slag in filling the furnace does not mention the analysis of the slag. A refractory slag would be objectionable, as it might drop into the hearth before it became free running, and would thus "build up" the hearth, instead of filling it with hot fluid slag. Not only must the hearth be hot when the slag strikes it, but the first slag should be hot and fluid. Evidently this was not the case in the blowing-in described by Mr. Firmstone, because "cinder showed at the tuyeres" within an hour after the stock had settled enough to start filling and the first flush was only four hours after the blast was put on all the tuyeres. The cast of white iron showed that the hearth was not hot enough or else the slag was wrong.

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The Effect of Alumina in Blast-Furnace Slags.*

BY J. E. JOHNSON, JR., ASHLAND, WIS.

(Cleveland Meeting, October, 1912.)

THE subject of blast-furnace slag is one which has had much consideration, particularly from the scientific standpoint, and several years ago technical literature contained many learned discussions on the oxygen-ratio of the acids and the bases, all on the assumption that acids and bases must be present in a ratio depending on molecular weight and chemical valency. Much work has also been done, sometimes in conjunction with the former, on the total heat and the temperature of initial fusion or softening of various slags.

A wider knowledge of the blast-furnace now enables us to say that for practical purposes this work has been of almost no value. It is now generally admitted that on the physical side the important point of a given slag is neither its total heat of fusion nor its softening-point, but its free-running temperature, since this is the critical temperature of the furnace.

This temperature bears no definite relation to the softening-temperature, as determined by Seger cones, for instance, because some slags melt relatively like ice, with a very short viscous-range, while others melt relatively like tar, with a very long viscous-range.

In one the difference between the softening- and the free-running temperature may be very small, as little as 100° F., while in another this difference may easily be 200° or 300°.

The importance of this difference is so great in practice as to destroy completely the value of exhaustive investigations, even such as those of O. Boudouard, published a few years ago.[1]

* Presented also at the Joint Meeting of the Institute with Section IIIa, Metallurgy and Mining, of the Eighth International Congress of Applied Chemistry, New York, September, 1912.

[1] It may not be amiss at this point to call attention to the excellent suggestion of Woolsey McA. Johnson, made several years ago, for determining the temperature at which different slags have the same practical fusibility. This suggestion

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When we come to the chemical side of the question we may as well dismiss the obscuration which comes from the introduction of oxygen-ratios and other abstruse theoretical considerations, and admit frankly that experiment on the furnace itself is the only safe foundation for practice or for a useful theory.

I am moved to lay stress on this point because in years past I was greatly puzzled and befogged by some of the older school of furnace-men, who emphasized the mysterious nature and difficulty of making slag-calculations, the fact being that, given the materials to be used and the slag to be produced, the college-student who could not be taught in a day to make the calculations, would have a poor chance of ultimate usefulness. The omission or concealment of the fact that the slag to be produced was known only by experiment and experience with the furnace itself, caused my mystification, and I therefore emphasize it here.

The object to be sought may be briefly stated as follows:

Given the materials to be used and the kind of iron to be made, to ascertain the slag which will produce the result with the least cost for coke and for flux, and will permit the greatest output. The three desiderata come in the order given.

We may in this discussion omit consideration of charcoal-practice because the slag-question is of minimum importance in that field, and the field itself is of minimum importance in the iron industry. Also because of my complete ignorance of that practice.

(This paper was written three years ago, but has remained unpublished until now. It so happens that in the interval I

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was briefly to make experiments in a small electrically-heated furnace, the bottom of which was a carbon slab with a small round hole through it. The temperature at which different slags would flow through this hole would furnish accurate information as to their relative practical fusibility.

To determine the proper size of the hole through the carbon block, I offer the further suggestion that the temperature of the slag flowing over the cinder-dam in standard furnace-practice be taken when the slag was about of normal composition. Then put some of this slag in the experimental furnace and heat it to the temperature observed. A few trials would determine the size of the hole through which it would flow at this temperature. This size of hole could then be used in future experiments with the certainty of obtaining results comparable with actual practice.

Investigations along these lines would be of real interest and value to furnace-men.

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have had extensive experience with charcoal-iron slags, but it is not considered desirable to introduce a consideration of that subject into this paper, because in the absence of the necessity for desulphurization, the problems offered by charcoal-slags are totally different from those of coke-slags. In a general way, however, it may not be amiss to state that observation of charcoal-slags has confirmed the position taken in this paper.)

Coming then to normal coke practice and remembering that generally the lowest fuel-consumption is obtained with the lowest critical temperature, the object desired may be stated more definitely as the production of the most fusible slag that will give the necessary desulphurization of the iron. This statement is perhaps subject to certain limitations from the fact that there are circumstances in which it is necessary to raise the critical temperature in order to enable the iron to absorb a large quantity of silicon. The extent of this limitation I am not able to state, but we will return to this subject later.

It is necessary to remember that while the effect of lime is to increase the basicity of slag and facilitate the removal of sulphur, its use is subject to grave limitations, for two reasons:

1. The lime being generally the major ingredient of the slag, to increase the percentage of the lime in the slag involves a more than proportional increase in the slag-volume; for instance, to increase its percentage from 50 to 55 requires an increase of more than 10 per cent. of the weight of the slag in the lime, or about twice as much in limestone.

2 (and much more important). The addition of lime raises the fusion-point of the slag very rapidly.

The desulphurizing effect of the slag is proportional, not only to its basicity, but also to its fluidity in an almost equal degree, so that while increased lime *per se* has a desulphurizing influence, this is, to an increasing extent, neutralized and finally reversed completely by its decreased physical activity.

This is well shown in basic practice, in which the highest sulphur iron is made, not with deficient lime, but with an excess (due to a change in ore or the like) so great that the heat available is unable to bring the very refractory slag to the free-running condition necessary for proper desulphurization.

In further illustration of the point, there are two distinct methods of making this kind of iron. The first consists in run-

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ning on a very calcareous slag, with which the silicon in the iron is kept down by the basicity of the slag in spite of the high temperature necessary to keep the latter fluid. The second consists in maintaining a slag of only moderate basicity and much lower fusion-temperature and keeping down the silicon by carrying a heavy ore-burden, which, of course, can easily be done with the lower critical temperature.

Furnaces running on the first plan always require more coke for basic than for foundry-iron, while those running on the second plan use less than for foundry-iron.

It seems to me practically certain, therefore, that there is a considerable range of lime-content in the slag for given conditions, in which the desulphurization of the iron is not appreciably affected, while the coke-consumption necessarily rises with the increase in the fusion-temperature consequent on the higher lime-ratio, as well as the increased slag-volume.

In the case of foundry-iron, the increase of lime may be positively detrimental because of the reduction of silicon in the iron. I have recently been informed of several cases in which the lime had habitually been carried too high, because this action was not understood. These cases of bad furnace-operation show the importance of this subject.

Effect of Magnesia.

In all theoretical slag-calculations the quantity of magnesia is multiplied by 1.4 to put it on the same basis as the lime, which, from the point of view of the oxygen-ratio and molecular weights, is perfectly correct. Practically, however, this does not work out. Furnace-men using a limestone with a variable content of magnesia have told me that careful observation had failed to show any difference whatever on the furnace whether the magnesia was high or low, and I have seen a furnace using half calcite and half dolomite, the calcite being relative impure and the dolomite very pure, put onto all dolomite, which should have made the slag excessively basic, on the basis of molecular weight, but which as a matter of fact showed no observable change.

The truth seems to be that magnesia is less active chemically than lime in about the same proportion that its molecular weight is less.

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An interesting proof of its small chemical activity is supplied by the manufacture of caustic soda from the carbonate. Quicklime is added to the carbonate solution and takes up the carbon dioxide from the soda; for this purpose magnesia is found to be perfectly inert and worthless. Sir Lowthian Bell's opinion that magnesia was so inert as to be useless for the removal of sulphur in the blast-furnace is well known, but this opinion is no more borne out by practice than that giving it a greater value than lime, the truth appearing to be that for furnace purposes, in all ordinary proportions, one is about as effective as the other.

It is very necessary, however, to recognize that the addition of a certain amount of magnesia has a marked effect in lowering the fusion-temperature of the slag, and is therefore of great use where calcareous slags are required, particularly in the manufacture of basic iron.

For practical purposes lime and magnesia may be considered as being of equal value, and hereafter in this paper "lime" will be used to mean the sum of lime and magnesia.

The Effect of Alumina.

The foregoing portion of this paper contains nothing essentially new and is intended as an introduction to the remaining portion, the substance of which seems to have been unknown to most of the furnace-men with whom I have discussed the subject.

The effect of alumina has been the subject of much discussion; some regard it as an acid, others as a base, while a few declare it can be made to act as a base or an acid almost at will.

It has seemed to me that under such circumstances the probability was that its action was neither acid or basic, but was perfectly neutral, simply a diluent affecting the viscosity of the slag to some extent, but, with a given ratio of lime to silica, not affecting its chemical nature at all.[2]

[2] This conception of a given constituent affecting the physical but not the chemical properties of slag, in which two kinds of properties are so closely interwoven, is difficult at first sight, but I have illustrated it for myself as follows. If we had an ordinary acid, such as hydrochloric acid, in one beaker, and a solution of caustic alkali in another, we could mix the two in any proportions and each addition of one or the other would result in a corresponding change in the acidity or basicity of the resultant solution, but if we added a considerable quantity of molasses, we should alter its viscosity and its "free-running temperature," without affecting its chemical properties in any way. The illustration is a homely one, but in my opinion the case is precisely parallel to that of slags.

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For several years the range of alumina in slag of which I had knowledge was so limited that I could not prove this contention, but very recently a furnace-man gave me complete information of a remarkable series of experiments he had carried out, in which the alumina in the slag had been as high as 39.5 per cent. with silica as low as 21 per cent. on individual flushes, and averaging for an entire day SiO_2, 24.7; Al_2O_3, 36.0 per cent.

The iron made was good Bessemer iron, about 2 per cent. of silicon, with sulphur about 0.023 per cent.

With this let us compare standard Lake-ore practice on basic iron, running about Al_2O_3, 13.5; SiO_2, 33 per cent.; and Virginia practice on the same iron, Al_2O_3, 6.5; SiO_2, 36 per cent.

It is unfortunate that the experimental run was on Bessemer instead of basic iron. The records of other days which show basic iron made are not so complete, but they indicate only a small difference in the slag; that shown, of course, being an increase in lime.

The coke-consumption of this slag was not materially different from what it was in standard practice; the slags were free-flowing, and did not have a noticeably higher fusion-temperature than ordinary.

Here, then, we have three cases, in all of which the coke is of about the same sulphur-content, the desulphurization of the iron is the same, the coke-consumption is no more different than would be accounted for by the different kinds of ore. The only difference of importance is the silicon in the iron, which is not sufficient to require a very great change in the slag-composition. We will refer to this condition later.

The amount of lime in these slags can be determined by subtraction, but it is necessary to remember that there is a small quantity of neutral material, CaS, FeO, MnO, etc., which may be taken at 3.5 per cent. in all cases as a close approximation to good practice. Following this procedure, the results shown in the first four columns of Table I. are obtained:

TABLE I.—*Composition of Slags.*

	1. Al_2O_3 Per Cent.	2. SiO_2 Per Cent.	3. CaO by Difference. Per Cent.	4. Neutral Substances. Per Cent.	5. Ratio $\frac{CaO}{Al_2O_3 + SiO_2}$	6. Ratio $\frac{CaO + Al_2O_3}{SiO_2}$	7. Ratio $\frac{CaO}{SiO_2}$
Virginia practice..	6.5	36.0	54.0	3.5	1.27	1.68	1.50
Lake-ore practice..	13.5	33.0	50.0	3.5	1.08	1.92	1.51
Special experiment	36 0	24.7	36.8	3.5	0.59	2.90	1.44

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Columns 5 and 6 present the ratios of lime to silica + alumina, and lime + alumina to silica. These ratios change from 1.27 to 0.59 in the first case, and from 1.68 to 2.90 in the second case.

No one can hope to show any relations between these ratios that can bear in any intelligible way on the variation of the alumina-content.

Column 7, however, shows the ratio of the lime to the silica, which is virtually a constant throughout. The result in the experiment is the lowest, and corresponds to the higher silicon in the iron.

If the lime were increased in this latter case about 5 per cent. of its own weight, this slag would then have identically the same ratio as the others, and would be just about as much "limier" as would permit the production of basic iron by the addition of a little more burden.

It may be claimed in opposition that the analyses chosen as representative of Virginia and Lake-ore practice have been taken with the object of showing the result desired rather than representing the typical slags. For the Virginia practice this certainly is not true, and if there be any error in the slag-analysis for Lake-ore, I do not know how to better it. Certainly no correction that could be made would fail to leave the lime-silica ratio infinitely nearer a constant than either of the others.

It may be well to reiterate here that such a comparison is only useful in the case of furnaces on a comparable basis in other respects, that is to say, working for the same degree of desulphurization and the same silicon, and using materials of about the same sulphur-content.

The question of the slag-volume also enters here, and the question whether a given slag-volume is as efficacious in the removal of sulphur when the alumina is high as when it is low. We know that if the slag exceeds a certain content of sulphur in ordinary practice, the desulphurization of the iron will be incomplete, and if the alumina be simply a diluent, the effectiveness of the slag might be diminished in proportion as the percentage of the alumina rose.

In spite of this theoretical consideration, the sulphur in the slags in the high-alumina experiments mentioned ran about

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1.75 per cent., with perfectly satisfactory desulphurization, while the relative slag-volume was little, if any, higher with alumina approaching 40 per cent. than with normal alumina.

On foundry-irons the ratio of lime to silica is, of course, lower than on basic. In Virginia practice the typical analysis would be about: Al_2O_3, 7; SiO_2, 41; CaO, 48.5; neutral substances, 3.5 per cent., giving a lime-silica ratio of 1.2, instead of 1.5 on basic iron.

A recent opportunity of discussing this subject with many furnace-men in all parts of the country has shown that virtually all slags for foundry-iron have a lime-silica ratio of between 1.2 and 1.4.

Foundry-iron practice is so much less definite than basic that the same exactness of ratio is hardly to be expected, and data are not at hand to determine a correct ratio definitely.

There seems to be a fairly well-defined opinion among the best informed furnace-men that alumina has the effect of making the slag viscous and stringy in a certain range, beginning about 16, reaching a maximum about 20, and disappearing between 25 and 30 per cent.

The data at hand do not permit of a definite statement on the subject, because the lime-silica ratio is not known for a sufficient number of cases, and if high alumina be coupled with a lime-silica ratio higher than normal, it is not fair to blame the alumina any more than the high lime for the result.

One case is on record in which the high alumina was coupled with excessively bad results, but it was also coupled with an extremely high-sulphur coke, which compelled the use of very high lime, resulting in an infusible slag. In such a case the high alumina may do no more than render stringy a slag that would be troublesome in any event.

A question of this character can only be properly settled by the use of an accurate pyrometer for determining the temperature at which different slags become free-running. The perfection of an accurate optical pyrometer which can easily be used to determine the temperature of the slag close to the furnace would make the investigation relatively easy.

The utility of such knowledge lies in enabling us to determine in advance the slag which will give satisfactory results in practice when a change in the materials used causes the alumina to exceed the ordinary limits.

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The decreasing grade of the ores available must result in the increasing use of many ores higher in alumina than those of present practice. If the laws governing its action are not known, bad results are bound to follow.

A furnace-man recently told me of a case in which the alumina had been regarded as an acid, and the slag had been calculated to give the same ratio of lime to silica + alumina as in standard practice, with the result that the furnace had been badly "bunged up" with lime.

Several furnace-men with whom I have talked had realized that alumina could not be properly regarded in this way, but only two have made definite statements that they regarded it as a neutral substance, and only one had reached the conclusion that a definite lime-silica ratio was the object to be attained. This was J. H. Frantz, General Manager of the Columbus Iron & Steel Co., whom I first met when this article was almost complete. Several years ago he had reached the opinion here maintained, and had prepared a table giving a lime-silica ratio of 1.5 with percentages of alumina varying from 8 to 30 per cent. He also told me that this view had been published several years ago, but he did not remember the name of the publication or of the author.

The fact that this publication as well as the principle seems to be entirely unknown to most furnace-men, and the increased importance attributed to the free-running temperature of the slag in recent years, are the only excuse for the present paper, in the face of prior publication of similar matter.

In conclusion, it seems worth while to point out that the increased viscosity accompanying high alumina may be of benefit in making foundry-iron. This was a subject to which both Mr. Frantz and I independently had given some consideration for several years.

That iron will not take up much silicon except at a relatively high temperature is well known, and as its melting-temperature is relatively low, its tendency is to run down out of the region in which it can absorb silicon into the crucible.

The fusion-temperature of slag is much higher than that of iron, and being more viscous, it acts as a retardant on the iron and delays its descent so that it can acquire the necessary temperature.

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A too-fusible slag does not perform this function properly, and I have personally seen a case in which no reasonable reduction of burden would raise the silicon to ordinary foundry-limits, but in which this was easily done by increasing the lime considerably beyond that necessary for desulphurization, in spite of the desiliconizing influence of the more basic slag. For the same reason it is more difficult to make foundry-iron when enough magnesia is present to increase the fluidity of the slag.

Now if alumina increases the viscosity of the slag without the accompanying desiliconizing influence of the limier slags of the same free-running temperature, then the introduction of the silicon into the iron should be facilitated by high alumina.

Mr. Frantz is of the opinion that this is true to some extent. My own experience does not cover the point.

R. H. SWEETSER, Columbus, Ohio:—I think that Mr. Johnson has opened up one of the most profitable discussions that could come to blast-furnace men, and I am very glad that he has come out so strongly against the old oxygen ratios. I will confess I never could understand them myself and never used them in practice. Now, I disagreed with Mr. Johnson strongly three years ago, when he first showed me his paper, and I thought that the question could be approached from the silica-alumina ratio side rather than from the lime-silica side. Eight years ago at "the Soo" we had an ore that absolutely refused to "behave in the furnace," and we could not use it; but going on the theory that all ores can be used in a blast-furnace if it is known how to handle them, we went over our records very carefully to find out when we could use ores of that nature successfully, and when we could not; and we found that the ore that was giving a great deal of trouble had a silica-alumina ratio that was very, very high—in fact, it had practically no alumina at all, and our slags had a very small percentage of alumina. I had experience in Tennessee with the Cranberry ore, which gave alumina in the slag down as low as 4.5 per cent., and I could not understand why the ore we had at "the Soo" could not be used as successfully. I located on a triaxial diagram the analyses of good and bad slags, and it seemed that the bad slags had a ratio of from five to eight of silica to

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one of alumina. Then I concluded that the silica-alumina ratio in our slag was too high, and that we could correct our trouble with this particular Mesabi ore by lowering the silica-alumina ratio. We made this change, and used the ore successfully. I have since learned that the "silica-alumina ratio" was partly right, and that Mr. Johnson's theory was partly right, for practical purposes; but neither explanation gives the true solution of the function of alumina in blast-furnace slags.

President Kemp spoke about a thing to-day that comes, I believe, nearer to the practical solution, and that is that the blast-furnace slags are mixtures of minerals, and that the working of slag depends on the melting-point of all the minerals that go to make up that slag.

I am indebted for a lot of information on this subject to Wilbur Stout, now with the Ohio State Geological Survey, who said that slags were really combinations of minerals, and that the melting-point would vary according to the number of minerals that were present, and that the melting-point would be governed greatly by the predominance of certain minerals. Mr. Frantz, who was mentioned in Mr. Johnson's paper, and I have worked over this a good deal together, and last week Mr. Frantz showed me a line across one of these triaxial diagram charts that seemed to explain the reason why alumina acts sometimes as a base and sometimes as an acid. We have actually replaced lime in our slags with alumina, and it seemed there was no change in the character of the slag; you could not tell, from its appearance, when there was 20 or only 14 per cent. of alumina present—the slags would look about the same to the blower, yet those slags varied greatly in the composition. I think there is a big field here if some of the scientists will take up this matter of slag analysis and determine these melting-points. The charts that have already been published have been prepared by men working in geology or in cement, but nothing has been done along the line of actual blast-furnace slags. I think it so necessary to explore the field that some one ought to take it up at once.

C. H. Fulton, Cleveland, Ohio:—A Scandinavian geologist, J. H. L. Vogt, has done a great deal of work on the mineral constitution of slags, and has correlated this work with the

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newer ideas of physical chemistry, constructing equilibrium diagrams such as are used for alloys, one of which was shown by Mr. Stoughton this morning. When the slags are looked upon from this view-point, we find that the entities that enter into a slag composition are minerals, and fortunately, most slags are composed of two minerals only. On this basis the action of alumina is rather simply explained, if we do not go too far into the theory of the subject. For instance, alumina may play the part of both an acid and a base. Whether it plays one or the other or both parts, depends entirely upon the mineral composition of the slag as a whole. If the chemical composition of the slag is such that the minerals augite, anorthite, melilite, or gehlenite may form, alumina acts as a base. If alumina is present in greater quantity than can be carried by the amount of these minerals formed, the excess performs the part of an acid. We find in certain slag compositions which contain alumina, that this will form spinel under the influence of the magnesia present. The practical point involved in the formation of spinel is this: the spinel (that mineral in which the alumina is the acid) has an extremely high melting-point, and it freezes out from the slag at such a high temperature that these slags are practically impossible in the furnace, especially when the spinel is present in large proportion. Widely varying percentages of alumina may be carried in slags, dependent upon what the rest of the base and silica contents are. What must be looked out for is that certain types of minerals are formed. If we know what these types are, and what their composition is, we have the key to the slag question. It is very difficult without charts to show the relation of mineral composition to chemical composition, but we find that for iron blast-furnace slags, in order to carry large amounts of alumina the silicate degree must be in the neighborhood of 1.00, with magnesia very low, otherwise such minerals as melilite and anorthite will not form, with the result that part of the alumina goes into the spinel form. There has been more work done on this subject than is indicated by what has been said to-night. It is mostly foreign work of scientific character. I had occasion recently to investigate the reverberatory furnace slags of a Western smelting company, to determine the influence of composition on melting-point, and the relation between melt-

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ing-point and fluidity-point, or "free-flowing temperature," as Mr. Johnson calls it. It is encouraging to note the results which may be obtained for slags by an investigation based on the equilibrium diagram method such as was before the Institute this morning for iron-carbon.

MR. JOHNSON: I desire to say that I did not mean to belittle the amount of work done; all that I meant to say was that it had all been useless. The scientific people have determined the softening-temperatures of these slags, but they do not interest us. The slag has to be run out of the furnace, primarily, and able to sulphurize the iron secondarily, or it is no good. What is needed is to know the temperatures at which these slags will run. We need an absolute viscosity test, to know that a slag of a certain viscosity will handle itself properly in the furnace at a certain temperature, and until we get results in that shape we cannot use them.

JAMES F. KEMP, New York, N. Y.:—Some very interesting work in this connection has been done in the Geophysical Laboratory of the Carnegie Institution of Washington. Arthur L. Day and E. T. Allen, of the staff, have been giving special study to the melting-points of the feldspars and other minerals.[3] The difficulty of determining the fusing-point in earlier years arose from the uncertainty attached to direct observations with the eye. A change of state from solid to liquid in a brightly glowing mass which sometimes passes through a viscous stage is not easy of accurate definition. Platinum weights were tried by previous investigators. When these sank, fusion was assumed. The later plan is to follow temperature changes with a thermo-couple. When the regularly rising temperature is suddenly accelerated by the latent heat, released by the change of state from solid to liquid, fusion is obviously taking place. Accurate and consistent records are thus obtained.

I was reminded, when Mr. Sweetser was speaking, of the work of Prof. J. H. L. Vogt, formerly of Christiania, Norway,

[3] A. L. Day and E. T. Allen, The Isomorphism and Thermal Properties of the Feldspars. Carnegie Institution, 1905. Summary in the *American Journal of Science*, Fourth Series, vol. xix., No. 110, p. 93 (Feb., 1905). Contributions on other mineral groups have since appeared from the Geophysical Laboratory.

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now of Trondhjem. Some hundreds of analyses of slags were compiled by Professor Vogt, and the minerals which had crystallized from them when such developed were also studied. Usually in the case of slag the fused mass chills so quickly that seldom do we get crystallized minerals. But occasionally, with a slow chill, a cavity forms in the midst of the pot, into which well-developed crystals project. By slow cooling also an interlaced mass of minerals like an igneous rock may develop; or a commingled mass of minerals and glass. As Professor Fulton has said, by comparing the known analysis of the slag with the results in crystallized minerals, Professor Vogt has brought out some very interesting relationships. The conclusions have also been of much importance in geological work because of the light cast upon the problems of the lavas and other igneous rocks. The study of Professor Vogt's paper I am sure would be of interest to many of our members engaged in iron smelting, since they could compare the composition of slags with their crystallized results and get some very interesting light. Professor Vogt's paper is published in German, and therefore is quite easy to read, if one has that language at command.[4]

JOHN J. PORTER, Staunton, Va. (communication to the Secretary *):—Mr. Johnson's paper is particularly timely and interesting because it deals with a phase of operating efficiency. The problem of high cost of living applies to manufacturing-plants as well as individuals, and its best solution is to be found in closer study of all the details of operation rather than in hasty and often haphazard expenditures for plant-equipment. If the best possible were always done with the equipment at hand, it would in many cases be found unnecessary to spend large additional sums, and in other cases would enable the money to be spent to far better advantage.

The subject of blast-furnace slags is a large one, and several

* Received Nov. 9, 1912.

[4] Professor Vogt has published a long series of papers, beginning as far back as 1884. The best one for the purpose above suggested is Beiträge zur Kenntniss der Gesetze der Mineralbildung in Schmelzmassen und in den neovulkanischen Ergussgesteinen. (Contributions to our knowledge of the laws of mineral formation in smelting products and in the later volcanic rocks. Christiania, Alb. Cammermeyer, Publisher, 1892. Pp. 271, pl. I., II.). This paper has been since followed

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points deserving discussion occur to me. I shall, however, confine myself to the one factor of formation-temperature which has been overlooked or ignored by Mr. Johnson, although it has, in my estimation, a decided bearing on fuel consumption.

My attention was first called to this point while studying conditions at certain Southern furnaces which experienced much trouble with viscid slag, and which were forced to use an excess of coke in order to correct this difficulty. The trouble had been attributed to high alumina on the basis of the theory discussed by Mr. Johnson, and this was apparently confirmed by the fact that substitution of other ores lower in alumina relieved the trouble. However, the range of alumina in the slag was only from 12 to 15 per cent., or about what is considered good practice with Lake ores, and a neighboring furnace working on a different ore-mixture, but making a slag of practically identical composition, experienced no trouble whatever.

The clue to the problem was given by some observations of slag-temperature taken by a Wanner optical pyrometer. It was found that at one of the furnaces in question, while the slag was excessively viscid, the temperature averaged over 100° F. lower than at another furnace working under similar conditions but using other ores not causing viscid slag. When excessive fuel was used to prevent viscosity of the slag the temperature became only normal, and there was no evidence that the slag was excessively superheated as one would naturally expect. The evidence, therefore, indicated that the viscosity was due to low temperature, rather than to high alumina or other defect in composition.

The difficulty was obviously associated with the use of high percentages of certain "hard red" or Clinton ores of the fol-

by two other monographs, Die Silikatschmelzlösungen: Part I., Ueber die Mineralbildung in Silikatschmelzlösungen, 1903; Part II., Ueber die Schmelzpunkt-Erniedrigung der Silikatschmelzlösungen, 1904. (Fused Solutions of Silicates: Part I., On the Formation of Minerals in Fused Solutions of Silicates; Part II., On the Lowering of the Fusing-Point in Fused Solutions of Silicates.) Both these papers were originally published in the Memoirs of the Academy of Sciences in Christiania, but were separately issued by the publishing house of Jacob Dybwad. Professor Vogt has continued to write in later years upon the questions of eutectics and related themes as applied to igneous rocks, but with less immediate reference to slags.

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lowing approximate composition: Fe, 37; SiO_2, 11; Al_2O_3, 6; and CaO, 11 per cent. Inspection showed that the silica was intimately associated mechanically with the lime and was not, as is the case with many of the Clinton ores, in the form of quartz grains. The substitution of either brown or "soft red" ore up to the extent of 50 per cent. of the burden caused the work of the furnace to become normal.

The theory which I evolved to account for these phenomena was as follows:

Slag is obviously formed in the molten state under the conditions prevailing in the blast-furnace, for it is only by the liquefaction or softening, and mutual solution of the constituents that they can come into the intimate contact necessary for chemical combination. After the slag has melted, however, it flows very quickly into the space below the tuyeres, and out of the zone of maximum temperature. It is generally admitted that slag can take up very little heat after melting, as it drops too quickly into the relatively cool space below the tuyeres.[5] Hence the temperature of the slag will be controlled very largely by its temperature of formation, and if this is low the slag may be insufficiently superheated above the melting-point, and hence viscid.

Now the temperature of formation is dependent very largely on the intimacy of contact of the slag-forming ingredients. If the gangue of the ore is chiefly in the form of grains of sand or layers of slate and the lime is added as lumps of limestone the points of contact will be few, the formation of slag will take place slowly, and temperature of formation will be high, because the constituents will descend far down in the bosh to a zone of high temperature before combination is effected. Moreover the slag when formed will be highly superheated, hence fluid, and the furnace will work smoothly, and have little tendency to hang in the boshes.

On the other hand, if the ingredients be in very intimate contact the formation-temperature will be much lower, possibly even below the melting-point, as in portland cement manufac-

[5] Of course there will be some superheating after melting, and this will be greater if the slag is viscid, as explained by Mr. Johnson. His idea of the utility of alumina in making high-silicon iron, is in line with what little information I have on the subject.

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ture. In the case described the gangue of the ore contains: SiO_2, 39.3; Al_2O_3, 21.4; CaO, 39.3; total, 100 per cent. Obviously this is a very fusible mixture and will, therefore, melt high up in the boshes, and, trickling down, will dissolve the lumps of lime and the coke ash before these have had a chance to become highly superheated. The slag not having reached a high temperature before melting will not be sufficiently superheated to be thoroughly fluid, and incidentally may cause more or less hanging on the bosh wall. Its temperature can, of course, be increased by increasing the fuel, but it is obvious that it will take more fuel to raise the temperature 100° when the molten slag is only in contact with the hot gases for a few moments, as in the last case, than when the solid slag constituents remain in the heated zone for a much longer time, as in the first case.

Undoubtedly it is possible to go too far in the matter of lessening the points of contact of slag constituents, and I certainly would not advocate increasing the size of the limestone, for example, or abandoning entirely the use of self-fluxing ores. On the contrary I believe that in only a few of the Clinton ores are the gangue constituents in such a chemical and mechanical relation as to produce this trouble. Where the silica is present either as grains of sand or as layers of slate, it is probably sufficiently segregated from the lime to prevent trouble.

The importance of slag-formation temperature has long been recognized in non-ferrous metallurgy, but so far as I know it has never before been discussed in its relation to iron blast-furnace practice, nor has the importance of the mechanical association of the slag constituents been adequately recognized.

W. McA. Johnson, Hartford, Conn. (communication to the Secretary *):—I read Mr. Johnson's paper with pleasure and interest, and I give below a *résumé* of the construction and use of my slag-tester which it alluded to in the first foot-note.

A slag is essentially a complex silicate of several bases. Calcium oxide is one of the most prominent bases in iron blast-furnace practice, while in lead or copper blast-furnace work, ferrous oxide is the most prominent. In the process of

* Received Nov. 30, 1912.

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cooling, the various silicates crystallize out similar to alloys, and as such, slags do not have a sharp melting-point.

In metallurgical work it is important that an accurate knowledge of the "running temperature" of the slag be had in order that a liquid slag be obtained under given metallurgical conditions. This is shown in the lead, iron, and copper furnace-practice, especially in the first-named. Likewise in zinc retort work this knowledge is essential so as to prevent the formation of slags. The formation-temperature varies greatly and is usually much below its melting-temperature, *i.e.*, the oxides of iron and calcium unite with silica in a dry way when intimately mixed and heated for a considerable interval of time. The question of time is an important point. Satisfactory data of this subject are not to be found in literature, for usually the uncertain "softening-point" was determined. Such results are not directly translatable into actual practice, unless subject to great corrections and mental reservations from the basis of practical knowledge.

While I was with the Orford Copper Co., in 1902, working on a small "nickel brick cupola," an opportunity was had to observe the process of slag-formation on the coke just as the cupola furnace was going out of blast at the end of the short weekly campaign. The observations were made through the small holes which appeared in the sides of the furnace.

While with the Lanyon Zinc Co., in 1903, I found that Acheson graphite was a most admirable material for making metallurgical research work. In that year an apparatus for the testing of the reduction-temperature of metallic oxides,[6] and a similar apparatus for the decomposition temperature of metallic sulphides, was devised. On the basis of this varied experience, when with the Lungwitz Reduction Co., at the instance of Mr. Fred W. Gordon, chief consulting engineer, the "slag-tester" described here, Fig. 1, was developed for the purpose of making determinations of "slag-running temperatures" that were accurate. The apparatus as designed proved neat and serviceable, and gave results directly applicable to large scale work.

Round bars of Acheson graphite 4 in. in diameter were cut in 3.5-in. lengths, with a narrow slot $\frac{5}{8}$ in. wide and 1.5 in. high

[6] *Transactions of the American Electrochemical Society*, vol. v., p. 211 (April, 1904).

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cut in the bottom. A machined drilled 1-in. hole was made in the top to the depth of 1 in. A $\frac{7}{32}$-in. hole was continued to the narrow slot.

A horizontal 1-in. hole was drilled above the line of the lower slot so as to receive a Le Chatelier pyrometer at a point as near as possible to the $\frac{7}{32}$-in. "dripping-hole." The upper hole was filled with small proportions of the formed slag or slag-making materials. The entire apparatus was placed in a

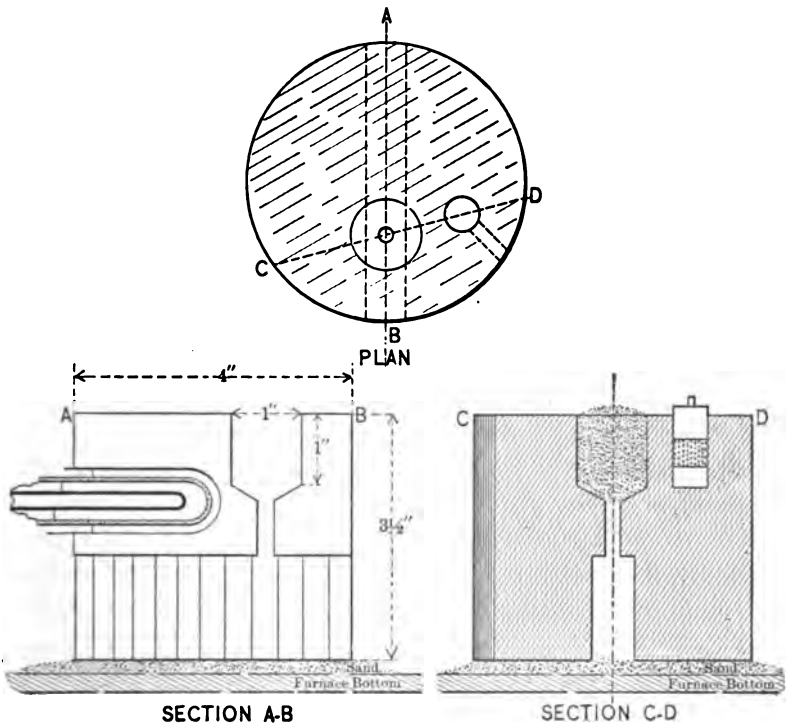

FIG. 1.—SLAG-TESTER. PLAN AND SECTIONS.

special oil-fired experimental reverberatory furnace, the hearth of which was covered with a layer of sand, and the pyrometer was inserted through a hole in one side of the furnace.

When the slag reached the "dripping-point," and dropped or ran through the slot, depending on its viscosity, it could be viewed directly through the slot by means of an observing-hole in the side of the furnace opposite the pyrometer-hole, and its dripping- or running-point determined. The accuracy of the

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results was increased by the fact that the pyrometer couple was so close to the slag at this critical temperature. Even though slag is a poor conductor of heat, Acheson graphite is so good a one that the temperature gradient is small.

A check on the pyrometer was secured by boring a second hole 0.5 in. in diameter in the top, in which 5 g. of zinc were put. The zinc was covered with charcoal and the hole plugged with a whittled graphite plug. A pyrometer was inserted through this hole and checked at the boiling-point of zinc, 920° C., shown by the characteristic snaky flame of burning zinc vapor coming through the pyrometer hole.

A much larger apparatus was also made, in which the slag was allowed to drip through a column of granulated coke 3 in. in diameter and 4 in. high. In this manner we measured the rate of reduction of reducible oxides, FeO and ZnO. Another apparatus was designed looking somewhat like an hour-glass, in which it was proposed to measure the rate of flow of slags through a small aperture at different temperatures.

Acheson graphite is the only commercial material to be used in making this apparatus, as fire-clay would be fluxed by the slag, and in using iron this would be oxidized, and the iron oxide thus formed would also be fluxed by the slag.

Several score of typical slags were tested in the apparatus, but the data of the test cannot be published, as the work was done in a cursory and fragmentary way. The results, if published, would be open to criticism that no safe or firm deduction could be made therefrom. By the use of a mental process somewhat better than semi-intelligent guessing, the results can be used by me.

This description of the apparatus has been rewritten (the original appeared in *Electrochemical and Metallurgical Industry*, vol. iv., No. 7, p. 262 (July, 1906), in the hope that if the proposal be thought a worthy one, an extended investigation of slags in this slag-tester be made by some one that can properly and efficiently do such work, *e.g.*, a professor in a university or some one connected with a national or State government department.

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The Utility of Efficiency-Records in the Manufacture of Iron.

BY JOHN JERMAIN PORTER, STAUNTON, VA.

(Cleveland Meeting, October, 1912.)

In taking up this subject it is first necessary to define our terms. Efficiency, in its engineering usage, means the ratio between actual and theoretical results, and efficiency-records thus involve the use of scientifically determined standards with which all results are compared. Scientific management is the methods used in increasing efficiency, and includes the use of efficiency-records. Efficiency-records do not alone constitute scientific management, but they are necessary to it, and any so-called system of management which is not based upon standards and standardization is fundamentally wrong.

An efficient management and efficiency-methods in management are not the same thing. There are some people who have an instinctive genius for management and accomplish results without exactly knowing how they do it. Such a manager may obtain a very high efficiency without the aid of scientifically determined standards. On the other hand, it sometimes happens that poor results will accompany the use of efficiency-records, due to a weak or indifferent management, which fails to make use of the information acquired. Efficiency-records are only a tool, and like other tools are worthless unless used.

The ordinary cost-sheet and production-statement do not go far enough. They tell what has been done, but give no information as to how well it has been done. It is true that the man who is familiar with the industry and the plant in question may draw inferences and form a general idea of good work or bad work, but it can only be a very general and hazy idea. Comparisons with previous records or records of other plants are unsatisfactory, since there is nothing to show the cause of the differences.

Efficiency-records supply this need. They compare actual

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results with standards representing the best possible, and thus indicate automatically how much improvement is possible. Further, a perfectly developed system of records should show the efficiency of each of the items making up the final result, indicate the cause of poor work, whether due to equipment, materials or something else, and thus enable the management to note immediately the source of the trouble.

Another valuable feature of efficiency-records is the information obtained incidental to the determination of standards. In some cases standards are obvious; in others they are very difficult to determine, and in still other cases it is impossible in the present state of our knowledge to make them scientifically accurate, and an empirical makeshift must be accepted.

There are very few plants where the attempt to determine standards for all details will not disclose some obvious economies, and this fact need be no reflection on the management. This point is perhaps best illustrated by means of examples and for this purpose I have first chosen a very insignificant detail in the operation of coking plants: the costs of the slats used in closing up the doors of box-cars. I am indebted to Charles Catlett for the suggestion of this illustration.

The cost of these slats per ton of coke shipped equals: cost per board foot $\times$ number of board feet used per car, and the number of board feet used per car is equal to the length of the strips, times the average width of the strips, times the number of strips used per car, plus any waste in cutting to length. This analysis calls attention to the importance of: 1, using the minimum length; 2, buying in such lengths as to permit using the minimum length without waste in cutting; 3, using the maximum permissible spacing. All of these points are simple enough in themselves, and yet, I will venture to say, are overlooked at many plants on account of their pettiness.

Another type of example is found in the determination of the standard blast-temperature in blast-furnace operation. Assuming that the ores used impose no limit to the temperature, the factors to be determined here are: the quantity of gas available for the stoves; the heating-value of the gas; the efficiency with which it is burned; the losses of heat in chimney gases, through radiation, etc.; and the amount of heat absorbed by the stoves and returned to the blast. Considerable investigation is necessary

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to secure this data, but in a number of cases where it has been done, unsuspected conditions and great losses, due to incomplete combustion and other causes, have been disclosed.

In the following pages are some examples of efficiency-reports as applied to blast-furnace operations. They represent detached portions of reports prepared in connection with the examination of several furnace plants, and are not a connected report on one plant. The way in which the data are presented is not the only way possible, and probably not even the best way. It would certainly be desirable to have somewhat different forms for reports which are to be made up at regular intervals as a matter of routine.

The operating efficiency of any industrial plant is measured in terms of the three factors: cost, output, and quality; hence the starting point in analysis of efficiency is Table I.

The factor of cost is next analyzed in Table II. Following, we have the prime factor of stock efficiency, which is covered in Tables III. to IIIe. The methods used in calculating quality efficiency of stock have been explained in a previous paper.[1] In the case of a concern producing its own ore, coke, and limestone, these reports should of course be supplemented by a set of mine-efficiency reports.

Tables IV. to IVc. are an analysis of the important item of fuel efficiency. This analysis is based on methods which I have previously described in my paper,[2] and which, while not perfectly accurate, have proved very satisfactory as a basis for efficiency-records. It is not absolutely necessary that standards be perfect. An approximate standard is much better than none at all, and in this case the accuracy of the standard fuel consumption is less important than the ability to analyze fuel requirements into prime elements, and thus show where improved work is possible.

Labor efficiency would properly be the next item to be analyzed, but as I have not had the opportunity of making a satisfactorily thorough study of this element I omit these tables. Standards for labor can only be determined by detailed study

[1] The Rational Valuation and Quality-Efficiency of Furnace-Stock, *Trans.*, xliii., 433 to 445 (1912).

[2] The Fuel-Efficiency of the Iron Blast-Furnace, *Trans.*, xlii., 191 to 221 (1911).

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according to the methods of Taylor[3] and Gilbreth.[4] I also omit for the same reason the tables relating to the efficiency of supplies, general expenses, etc.

Tables V. to Vc. deal with the factor of output efficiency. The output equation is self-evident and calls for no comment. In regard to the rate of blowing, however, there are two cases to be considered. First, the plant having insufficient blowing capacity: In this case the standard rate of blowing is of minor importance, since it will not be attained. What is important is the analysis of blowing efficiency, as given in Table Va. and supplemented by an investigation of boiler and engine efficiency. The second case is that of those plants amply supplied with blowing-power. In this event Table Va. is of small importance, and what is necessary is to determine the maximum rate of blowing desirable under the given conditions.

Unfortunately, there is no known theoretical basis for such a calculation, and the only way of determining an accurate standard is by direct experiment, although of course some inferences can be drawn from experience elsewhere. The perfect solution of this problem would be of great value, since it would enable one to plot output against fuel cost, and to calculate at once the most profitable rate of driving for each price of iron.

Table VI., quality efficiency, needs some explanation. It is assumed in the first place that quality efficiency consists partly in making the kind of iron desired, whether this be of low or high grade. In the second place, I have assumed that for every grade there is an ideal analysis which in the long run it is advantageous to maintain. See Table VIa. Possibly no higher price may be obtained for a No. 2F running 0.03 per cent. of S. than for one running 0.049 per cent., but the furnace which habitually makes low-sulphur iron acquires a reputation which greatly facilitates sales in dull times and is a very real asset.

The relation of percentage of silicon and sulphur to silicon and sulphur efficiency is entirely a matter of assumption. For my own work I have constructed diagrams giving an empirical relationship which have been satisfactory as far as I have used

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[3] Shop Management, by F. W. Taylor, *Transactions of the American Society of Mechanical Engineers*, vol. xxiv, pp. 1337 to 1456 (1903).

[4] *Motion Study*, by Frank Gilbreth, D. Van Nostrand Co. (1911).

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them. The value of quality standards does not consist, as in some cases, in analyzing the factors affecting quality, but only in giving a reliable index figure showing the perfection of the work done.

Efficiency-records should be especially useful in giving the chief executive better control of details and in promoting co-operation between departments.

As an example of what I mean, consider Tables IV., IVb. and IVc. It is shown there that one of the chief reasons for the high fuel consumption is the very high ash of the coke. The furnace superintendent knows this, but it is beyond his control, and he contents himself with an occasional statement that "we could do a whole lot better if we had decent coke." The superintendent of the coke-plant does not understand and is not particularly interested in the troubles of the blast-furnace man. His problem is to keep down costs and to keep up the output, and to him quality is a minor matter.

The chief executive, whose duty it is to co-ordinate the work of the different departments, has no definite information on which to act. He may or may not see an occasional analysis showing high ash, and he may receive an occasional complaint from the furnace superintendent. There are, however, no figures to prove that the matter is serious, nothing to show that better can be done, and it is not called forcibly to his attention.

On the other hand, with efficiency-records the facts in the case are presented so squarely, and the necessity for special effort is made so apparent, that it is hardly conceivable that some action would not be taken.

TABLE I.—*Total Operating Efficiency.*

	Standard.	Actual.	Efficiency.
			Per Cent.
Cost of manufacture per ton	$9.05	$9.94	91.0
Output, tons per day	175.	158.	90.3
Quality efficiency			93.0
Total operating efficiency			76.4

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TABLE II.—*Cost Efficiency.*

	Standard.		Actual.		Efficiency.
	Per Month.	Per Ton Iron.	Per Month	Per Ton Iron.	Per Cent.
Tons used, ore	10,716.00	2.090	7,847.00	2.140	97.5
Tons used, coke	7,061.00	1.390	5,742.00	1.566	88.8
Tons used, flux	5,054.00	0.995	3,850.00	1.050	94.8
Cost.					
Ore	$19,980.12	$3.930	$14,773.61	$4.031	97.6
Coke	18,149.88	3.570	14,788.17	4.035	88.8
Flux	3,086.00	0.607	2,389.88	0.652	93.1
Wages	4,824.72	0.949	4,258.36	1.161	81.6
Salaries	400.00	0.079	355.00	0.097	81.5
Supplies	778.00	0.153	585.00	0.159	96.3
Coal	310.00	0.061	374.40	0.102	59.8
Relining	1,779.40	0.350	1,178.80	0.350	100.0
General expenses	1,098.14	0.216	1,098.14	0.300	72.5
Total cost	$50,406.26	$9.915	$39,801.36	$10.887	91.0

TABLE III.—*Stock Efficiency.*

	Quality. Value Per Ton.			Cost Per Ton.			Total Efficiency.
	Standard.	Actual.	Efficiency	Standard.	Actual.	Efficiency	
			Per Cent.			Per Cent.	
Ore mixture	$2.13	$2.02	94.8	$1.88	$1.88	100.0	94.8
Stone mixture	0.64	0.61	95.3	0.61	0.62	98.4	93.7
Coke mixture	2.61	2.55	97.6	2.57	2.57	100.0	97.6

It is not correct to judge of the efficiency of furnace stock solely on the basis of the quantity used and the price paid (*i. e.*, on the basis of cost per ton of iron), since its quality affects fuel consumption and output, and, through these, other items of cost. The figures given above for value per ton take all these factors into consideration, and give the true smelting value irrespective of cost.

The constituents of the standard and actually used mixtures are given in Table III*a*.

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TABLE III*a*.—*The Mixture.*

Ore Mixture.			Flux Mixture.			Coke Mixture.		
Ore.	Standard.	Actual.	Flux.	Standard.	Actual.	Coke.	Standard.	Actual.
	Per Cent.	Per Cent.		Per Cent.	Per Cent.		Per Cent.	Per Cent.
No. 1	80.0	80.0	No. 1	25.0	40.0	No. 1	100.0	100.0
No. 2	15.0	15.0	No. 2	75.0	60.0	.....	.....	.....
No. 3	1.0	1.0	.....	.....	.....	.....	.....	.....
No. 4	3.0	3.0	.....	.....	.....	.....	.....	.....
No. 5	1.0	1.0	.....	.....	.....	.....	.....	.....
	100.0	100.0		100.0	100.0		100.0	100.0

The analysis of the mixture is given in Table III*b*.:

TABLE III*b*.—*Analysis of Mixture.*

	Iron.	Insoluble.	Bases.	Carbonates.	Carbon.	Ash.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Standard ore mixture....	44.30	20.15	.....	.....	.....	.....
Actual ore mixture.....	43.30	20.77	.....	.....	.....	.....
Standard flux mixture....	.....	1.17	50.51	97.13	.....	.....
Actual flux mixture.....	.....	2.13	50.59	96.20	.....	.....
Standard coke mixture..	.....	.....	.....	.....	92.00	6.30
Actual coke mixture.....	.....	.....	.....	.....	92.00	6.30

TABLE III*c*.—*Efficiency of Individual Ores.*

	Iron.	Insoluble.	Bases.	Relative Value.[a]	Quality Efficiency.	Cost Per Ton.	Relative Desirability.	Efficiency.
	Per Cent.	Per Cent.	Per Cent.		Per Cent.			Per Cent.
No. 1 Standard.....	43.50	21.60	.....	$2.01	.....	$1.90	106.	.....
No. 1 Actual.....	42.50	22.40	....	1.90	94.6	1.90	100.	94.6
No. 2 Standard.....	49.00	11.00	.....	2.81	.....	1.80	156.	.....
No. 2 Actual.....	47.93	11.46	.....	2.70	96.1	1.80	150.	96.1
No. 3 Standard	50.75	9.22	.....	3.01	.....	2.00	150.	.....
No. 3 Actual.....	50.75	9.22	.....	3.01	100.0	2.00	150.	100.0
No. 4 Standard.....	42.00	28.00	.....	1.69	.....	1.65	102.	.....
No. 4 Actual.....	41.33	28.50	.....	1.62	96.0	1.65	98.	96.0
No. 5 Standard.....	40.50	18.50	.....	1.85	.....	1.88	98.	....
No. 5 Actual.....	39.69	19.07	.....	1.76	93.4	2.07	84.	85.8

[a] Value of iron unit, $0.0896.

Table III*c*. was followed by a statement of the reasons for choosing the standard mixtures and the standards for quality and cost. Also some data as to the relative desirability and value of other ores available but not used. This section is omitted for the sake of brevity and to avoid disclosing the identity of the ores.

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TABLE IIId.—*Efficiency of Individual Fluxing Stones.*

		Insoluble.	Carbonates.	Relative Value.	Quality Efficiency.	Cost Per Ton.	Relative Desirability.	Efficiency.
		Per Cent.	Per Cent.		Per Cent.			Per Cent.
No. 1	L. S. Standard	3.00	95.50	$0.57	.....	$0.65	88.	.....
No. 1	L. S. Actual	4.50	94.00	0.50	87.7	0.65	77.	87.7
No. 2	Dol. Standard	0.55	97.67	0.68	.....	0.60	113.	.....
No. 2	Dol. Actual	0.55	97.67	0.68	100.0	0.60	113.	100.

TABLE IIIe.—*Efficiency of Individual Cokes.*

		Fixed Carbon.	Ash.	Factor of Physical Structure. a	Relative Value.	Quality Efficiency.	Cost Per Ton.	Relative Desirability.	Efficiency.
		Per Cent.	Per Cent.			Per Cent.			Per Cent.
No. 1	Standard	92.0	6.3	0.95	$2.61	.....	$2.57	101.5	.....
No. 1	Actual	92.0	6.3	1.00	2.55	97.6	2.57	99.4	97.6

a For the best structure this factor is 0.5 ; for the poorest 1.0.

TABLE IV.—*Detailed Analysis of Fuel Efficiency.*

	Standard.	Actual.	Efficiency.	Remarks.
			Per Cent.	
Blast-temperature, ° F.	1,250	1,153	92.3	
Moisture in blast, g.	5.0	6.0	83.3	Steam leaks.
Critical temperature	2,700	2,700	100.0	
Heat available per lb. of C.	2,040	1,825	89.5	
Slag per ton of iron, lb.	1,960	2,200	89.1	Chiefly coke ash.
Silicon in iron, per cent.	2.75	2.11	130.0	Efficiency from fuel standpoint only.
Heat needed in hearth	3,210	3,270	98.2	
Rate-driving factor	1.0	1.0	100.0	
C in hearth, T. I., lb.	1,574	1,717	91.7	
Carbonates per T. I., lb.	1,800	1,900	94.7	
Size-factor carbonates	0.9	0.9	100.0	
Fuel-factor	0.8	0.9	89.0	
Carbonate loss, lb.	155	184	83.8	
Reducibility of ore	0.3	0.3	100.0	
Ore loss, lb. C	168	189	89.0	
Total loss C. in shaft	323	373	86.6	
Carbon in pig iron	90	90	100.0	
Total carbon needed, T. I.	1,987	2,180	91.1	
Fixed carbon in coke, per cent.	83.0	78.0	94.0	
Coke needed in furnace	2,394	2,790	85.7	
Loss as braize	96	114	84.2	
Total coke per ton of iron	2,490	2,904	85.6	

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TABLE IV*a*.—*Hot-Blast Stove Efficiency.*

	Standard.	Actual.	Efficiency.
			Per Cent.
Gas produced per lb. of coke, cu. ft. [a]	66.5	66.5	.....
Gas lost, per cent	1.0	1.0	100.0
Gas to boilers, per cent	76.5	76.5	100.0
Gas to stoves, per cent	22.5	22.5	100.0
B.t.u. per cu. ft. gas	100.	108.	108.0
B.t.u. supplied stoves per lb. of coke [a]	1,495.	1,616.	108.0
Cu. ft. blast at 32° per lb. coke [a]	49.3	49.3	100.0
Heat supplied stoves utilized in blast, per cent.	66.2	55.8	84.3
B.t.u. in blast per lb. coke [a]	990.	887.	89.6
Temperature of blast	1,005.	900.	89.6

[a] Refers to lb. coke entering furnace.

TABLE IV*b*.—*Analysis of Heat Losses of Stoves.*

	Standard.		Actual.	
	B.t.u. Per Min.	Per Cent.	B.t.u. Per Min.	Per Cent.
Radiation loss	98,400	15.6	85,280	13.9
Loss due to changing stoves	1,300	0.2	1,100	0.2
Loss due to incomplete combustion	.....	00.0	97,500	15.9
Loss sensible in chimney gases	113,000	18.0	88,200	14.2
Total loss	212,700	33.8	272,080	44.2
Heat in blast	415,300	66.2	340,200	55.8
Total heat supplied	628,000	100.0	612,280	100.0

TABLE IV*c*.—*Manner in Which Coke Is Used.*

	Coke Per Ton Iron.				
	Standard.	Actual.	Loss.	Loss.	Efficiency.
	Lb.	Lb.	Lb.	Per Cent.	Per Cent.
Constant for furnace	568	617	29	7.0	95.4
To smelt slag	581	700	119	28.7	83.0
To reduce silicon	405	400	—5	—1.2	101.2
Total C used in hearth	1,574	1,717	143	34.5	91.8
Dissolved by CO_2 of carbonates	155	184	29	7.0	84.3
Dissolved by CO_2 from reduction of ore	168	189	21	5.1	88.9
In pig iron	90	90	0	0.0	100.0
Total C used in furnace	1,987	2,180	193	46.6	91.9
Ash and volatile matter	407	610	203	49.1	60.8
Braize	96	114	18	4.3	84.2
	2,490	2,904	414	100.0	85.6

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TABLE IV*d*.—*Sources of Slag.*

	Slag Per Ton Iron.		Excess Slag.		
	Standard.	Actual.			Efficiency.
	Lb.	Lb.	Lb.	Per Cent.	Per Cent.
Ore No. 1	335	351	16	6.7	95.4
Ore No. 2	1,033	1,033	—	——	100.0
Total ore	1,368	1,384	16	6.7	98.9
Limestone	92	108	16	6.7	85.2
Coke (less braize)	500	708	208	86.6	70.5
	1,960	2,200	240	100.0	89.1

TABLE V.—*Output Efficiency.*

	Standard.	Actual.	Efficiency.
			Per Cent.
Minutes per day blast is on	1,436.	1,426.4	99.3
Coke burned per min., lb.	280.	227.	90.0
Yield of ore plus flux, per cent.	32.40	31.44	96.9
Burden ratio, corrected for braize	2.82	2.63	93.4
Output per day, tons	164.0	118.3	72.2

$$\text{Output per day} = \text{number minutes} \times \text{pounds coke burned per minute} \times \text{yield ore plus flux} \times \frac{\text{Ore plus flux}}{\text{coke}} \times 2240.$$

Coke burned per minute is a measure of rate of blowing and is equal to:

$$\frac{\text{Rev. per min.} \times \text{per cent. of air reaching furnace} \times \text{cu. ft. of air per rev.}}{56 \times \text{temperature correction factor} \times \text{per cent. fixed carbon in coke.}}$$

TABLE V*a*.—*Efficiency of Blowing.*

	Standard.	Actual.	Efficiency.	Remarks.
Rev. per min.	64.5	62	96.2	Poor boiler practice.
Per cent. of air reaching furnace	62.0	55.2	89.0	Valves and pipes leaky.
Cu. ft. of air per rev.	385.	385.	100.0	
Cu. ft. of air per lb. carbon	60.	60.	100.0	56 × temp. cor. factor.
Per cent. fixed carbon in coke	92.0	92.0	100.0	
Lb. coke burned per min.	280.	227.	90.0	

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TABLE V*b*.—*Causes of Loss of Time.*

	No. of Stops.	Min. Per Stop.	Total Min.	Total Per Cent.
Casting	122	2	244	58.0
Cleaning downcomer	1	128	128	30.0
Changing tuyeres	2	18	36	9.0
Changing monkey	1	9	9	2.0
Looking for water leak	1	3	3	1.0
	127		420	100.0

TABLE V*c*.—*Distribution of Output Loss Over Plant, Stock, and Operation.*

	Tons Per Day.	Total Loss, Per Cent.
Loss of output due to plant	30.2	66.1
Loss of output due to stock	8.5	18.6
Loss of output due to operation	7.0	15.3

TABLE VI.—*Quality Efficiency.*

Tons made	3,368.	
Tons of grades desired	3,368.	All f'dry grades acceptable.
Grade efficiency, per cent	100.	
Silicon efficiency, per cent	93.	Average of efficiencies for each cast separately.
Sulphur efficiency, per cent	96.	
Analysis efficiency, per cent	89.2	Product of silicon and sulphur efficiency.
Total quality efficiency, per cent	89.2	Product of grade and analysis efficiency.

TABLE VI*a*.—*Ideal or Standard Analysis upon which Analysis Efficiency is Based.*

Grade.	No. 1F. F'dry. Per Cent.	No. 2F. F'dry. Per Cent.	No. 2P. F'dry. Per Cent.	No. 3 F'dry. Per Cent.	
Silicon	3.00	2.50	2.00	1.50	or over.
Sulphur	0.02	0.03	0.04	0.05	or under.

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The Manufacture of Coke.

BY WILLIAM HUTTON BLAUVELT, SYRACUSE, N. Y.

(Cleveland Meeting, October, 1912.)

COKE is the product of dry distillation of bituminous coal, by which the volatile matter is driven off, producing a hard body of cellular structure. Not all bituminous coals will coke, and there has been much discussion regarding the substance which is present in coking-coals to distinguish them from dry coals. Professor Lewes says that the bituminous matter in coal is largely derived from the spores of fossil mosses. Resins are present in all soft coals, and contribute largely to the cementing of the coke. The resinates and hydrocarbons begin to distil out at moderate temperatures and leave the pitchy residual, which at 500° C. forms a mass already well caked together, if the proportion of humus residual is not too great. This coke is soft, but if the heat be raised to 1,000° C. the pitch decomposes further, leaving carbon, and Professor Lewes is convinced that the coking bond is due to these liquid products which distil off, leaving pitch, which then carbonizes and binds the mass into coke.

Professor Parr concludes that the structures of the organic compounds of the coal which furnish the cementing-material for coke, and which are apparently attacked by oxygen, have not been determined and seem to vary somewhat in different types of coal. However, they yield, on oxidation, humic acids of varying composition which decompose into powdery residues. Because of the complex nature of these substances and the difficulty experienced in isolating and identifying them, the theory of coking is still an open problem and the explanations advanced are largely hypothetical.

In the coking process the volatile matter driven off in the oven consists mainly of tar, ammonia, and gas, familiarly known as by-products. Both the quality and quantity of coke and by-products vary greatly with the composition of the coal.

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In America, coals similar to those of the Pocahontas region, containing as low as 16 per cent. or less of volatile matter, stand at one end of the list. In Europe, some coals are coked which contain not more than 13 per cent. of volatile matter. These produce the maximum yield of coke and the minimum yield of by-products. At the other end of the list are the gas-coals, containing as much as 38 or 40 per cent. of volatile matter, and yielding correspondingly small amounts of coke. The development of the best cell structure is of the highest importance in the production of metallurgical coke. A well-developed cellular structure presents a large surface to oxidation by air, producing rapid combustion, while the strong cell-wall prevents crushing and maintains an open fuel-bed. A soft thick cell-wall is objectionable in blast-furnace coke, since it permits the oxidizing action of CO_2 gases in the upper part of the furnace, by which the coke is wasted and the furnace-top temperature is increased. In foundry-work it is more desirable to have a coke with a softer cell-wall, smaller cell space, and a strong, tough structure. This permits the air to burn the coke immediately to carbon dioxide, with production of the highest temperature in the melting-zone of the cupola; and the weight of the charge is easily supported. Thus present practice seems to indicate that somewhat different treatments are required for producing furnace- and foundry-coke.

Coals high in oxygen are usually deficient in coking-qualities, and either will not coke at all, or produce a coke of weak, friable structure. But many coals, as will be shown later, which yield poor coke under ordinary conditions, may, by proper treatment, be made to produce good metallurgical coke. Coal when stored absorbs oxygen, sometimes with serious effect on its coking-qualities. The chief reason for this appears to be the effect of the oxygen on the resinates, which Professor Lewes says are thereby converted into humus.

The history of coke making has been discussed at length in the technical publications, by me[1] and others, and any reference to it here would be unnecessary repetition. The same statement applies to the earlier forms of bee-hive and by-product ovens, and the development from these earlier forms to modern

[1] *Mineral Industry*, vol. iv., p. 215 (1895).

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types. The term by-product oven is commonly applied in this country to the retort oven, although, of course, the by-products may or may not be recovered from the gases from retort ovens, depending upon market conditions. All the retort oven-plants in the United States have by-product recovery-apparatus. In this paper, the term by-product oven will often be employed, as being a more familiar term, in referring to the retort oven.

For many years the bee-hive oven was the standard for coke making, and has remained the standard in England and America long after its practical disappearance from continental Europe. There are several reasons for this. In England, the Durham coal, and in America, the Connellsville coal, are remarkably well-suited to the conditions of the bee-hive oven, so that the best quality of coke could be produced without difficulty, and with relatively good economy. In America, the bee-hive oven had certain economic advantages. It is quickly built, and at relatively low cost, and the labor required for operation is of low grade. It can be put out of run at relatively small loss during periods of industrial depression, and can be started up again with ease after a shutdown. These qualities adapted it to the fluctuating and changing conditions which for many years surrounded our iron industry. But the iron industry has become more stable; and, moreover, the coals best adapted to the bee-hive oven are growing scarcer, especially with the exhaustion of the Connellsville field. The U. S. Geological Survey reports that in 1911 the total coke production in the United States showed a decrease of about 15 per cent. while the output of by-product coke increased 10 per cent. Compared with five years ago the following are the figures of production:

	By-Product.	Bee-Hive.
1907	5,607,899	35,171,665
1911	7,847,845	27,207,517

This shows an increase in by-product coke of 40 per cent., and a decrease in bee-hive coke of 22 per cent., so that in 1911 the tonnage of by-product coke was 28 per cent. of the bee-hive tonnage. By-product plants now under construction and in contemplation will greatly increase the present output. In England the relative growth of the by-product oven has been even more rapid during the last few years.

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The generic differences between the structure of the bee-hive and by-product or retort oven need not be again described here. It is sufficient to say that the bee-hive process consists essentially in the heating of the coal with controlled admission of air to the coking-chamber, so that the heat necessary for the distillation of the volatile matter is produced by combustion within the oven-chamber. In the by-product oven the process is a true dry distillation. No air is admitted to the chamber, and the heat necessary for the distillation is supplied through the chamber walls.

The bee-hive oven has been carried to its highest perfection in this country; drawing- and loading-apparatus has been applied to it, to reduce the arduous labor of the old hand-method; and modifications of the original bee-hive shape have been successfully introduced, which add materially to the economy of operation, while maintaining the principles of carbonization as employed in the standard bee-hive.

The bee-hive oven is usually located at the coal-mine, and the coal is in very many cases charged direct to the oven without weighing, so that it is difficult to ascertain the exact yield of the oven; but except when coals are used which are especially adapted to the bee-hive process, the coke yields are considerably below the theoretical, and, of course, all the by-products are wasted. These facts, except in unusual cases, make it impracticable to locate bee-hive ovens away from the mines. On the other hand, the by-product oven is usually, in this country at least, located at the point of consumption of the coke, or at some center of distribution. This freedom of location of the by-product oven has a number of advantages. Although it entails freight charges on from 1.2 to 1.4 tons of coal for every ton of coke produced, yet coal usually carries a lower rate than coke, and is more easily transported, not requiring special cars and not being injured by handling. In some cases it can be shipped by water, with material saving in freights, and with proper care it can be stored at the plant in almost any quantity without material deterioration. This permits a blast-furnace plant, having its own coke-ovens at the furnace, to possess an assured supply of coke, independent of weather or shipping-conditions. It is quite common for by-product oven-plants to accumulate a stock of from one to eight

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months' supply of coal. The by-products produced are much nearer their market, and the gas is often available for industrial uses or for municipal lighting. The plant is nearer a supply of diversified labor, which is an advantage in the more varied processes of by-product oven-operation. An important advantage of locating the oven-plant at the point of consumption is, that it permits a convenient assembling of several kinds of coal at the ovens. This mixture of coals is often a great advantage, since it permits the best quality of coke to be produced, when the coke made from any one of the coals alone might be of inferior quality, or perhaps not at all adapted to the market requirements.

There has been much discussion regarding the relative cost of bee-hive and by-product oven-plants. It is difficult to make an exact comparison, since the functions of the two types are quite different. For example, in many cases the cost of a by-product oven-plant includes a large expenditure for coal-storage of several hundred thousand tons, in order to take advantage of water freight-rates. On the other hand, bee-hive plants are often built in connection with coal-mining plants and utilize a share of power-plant, water supply, etc., without having these necessary adjuncts included in their capital account. Careful inquiry into the actual cost of bee-hive oven-plants shows that to build such a plant complete in every respect, and in the best manner, including all the equipment besides the ovens and their immediate appurtenances, such as electric power-plant, water-supply, railroad-approaches and sidings, coal-handling, etc., would require an expenditure of about $950 per oven. The U. S. Geological Survey report for 1911 gives the tonnage of coke produced per active bee-hive oven at 466 tons per annum. This figure is doubtless low for the best bee-hive ovens; 675 to 700 tons per annum will perhaps represent fairly the average output per oven of a modern bee-hive plant. An average of these figures gives a plant-cost of $1.38 per ton of coke produced per annum. A by-product oven-plant of, say, 80 ovens, complete in every respect, and built in the best manner, would produce, say, 425,000 tons of coke per annum from an average coal, and would cost, say, $1,100,000. Of course, this figure would be varied by local conditions. This is equal to $2.58 per ton of coke per annum. On the basis of

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these figures a by-product plant costs 86 per cent. more than a bee-hive plant.

The owner of a small acreage of coking-coal might perhaps well hesitate before making the larger investment, as compared with a simple plant of bee-hive ovens, but, as a general proposition, there hardly seems to be a question as to which style of oven is the better investment. Much of the additional cost of the by-product plant per unit of coke produced is, of course, due to the installation of the apparatus and buildings for the recovery and treatment of the by-products. Other important items are provisions for coal- and coke-storage, thereby assuring uniformity of operation and greater ability to maintain uniform deliveries. Moreover, a by-product plant is built for a longer life than a bee-hive plant. Eight years may perhaps be considered as a satisfactory life for a bee-hive oven-plant, while double this term, or 16 years, would be several years within the life of a well-built by-product plant.

Market conditions have not been such in this country as to justify construction of retort oven-plants without the recovery of by-products, but we may assume a non-recovery oven-plant built for operation in the Pocahontas region, for example. The yield of coke from Pocahontas coal averages certainly not more than 60 per cent. in a bee-hive oven. In a retort oven the yield is more than 80 per cent. Such a plant without by-product recovery costs not much more per ton of product than an equally well equipped bee-hive plant, and the labor of operation under such conditions would be as low, if not lower. A plant of retort ovens in place of bee-hives in that field would, therefore, mean that at practically the same expenditure the owner of a coal property would, on account of the greater yield of coke, increase the life of his coal-field 33 per cent. with a given output of coke. Or the value of the property, based on the selling-value of the coke therefrom, would be increased 33 per cent.

The present development of the art, however, has brought the Pocahontas coal to the ovens located at the point of consumption, and the mixing of low and high volatile coals has grown in favor, especially during the last few years.

The cycle of operations in connection with the by-product oven has been described several times in technical papers, and

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need not be repeated here. It is well recognized that the operation of these plants calls for technical skill of high order, as the several processes include metallurgical, mechanical, and chemical engineering, and the success and economy of the operation are largely dependent upon the accurate control of a number of quite different and independent conditions. There has been great progress in the design of the ovens and recovery-apparatus. As might be expected, America now leads Europe in output of plant, size of oven, and rate of coking. In the early days of the by-product oven in America, the capacity of the oven was about 4.4 tons of coal per 24 hr., and 25 ovens were considered about the right number for one crew of men. Modern ovens have a capacity of as much as 20 tons of coal per oven per day, and by the introduction of more machinery and more efficient design, the number of ovens handled per man is also increased greatly. The rate of coking is one point in which American practice has gone ahead of Europe.

Retort ovens have been built of various widths from about 14 to about 30 in., to suit various ideas of the designers, and various coals; but, within limits, the rate of coking per inch of coal does not vary materially with the width of the oven. Not many years ago the best rate of coking was about 1 in. of oven width in 90 min.; that is, a 16-in. oven was coked in 24 hr. To-day there is more than one type of oven which is coking regularly at the rate of from 50 to 55 min. per inch of oven width. This increased rate was made possible, partly by better control of the heating-systems, and partly by the adoption of silica brick in the oven construction. Silica brick has been used in bee-hive oven construction very generally for a number of years. In retort ovens it was first used in the Otto-Hoffmann oven-plant at Johnstown in 1899, and is now the standard material in America for retort-oven construction.

During the past few years special study has been given to the heat economy of the retort oven and material advance has been made. From most American coals, more gas is produced than is necessary for heating the ovens. The conditions under which it is to be operated have an important bearing on the design of an oven-plant. It may be said that there are three general conditions. First, where there is not sufficient market to dispose of the surplus gas. In this case the design of oven

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should be very simple. Any complication of the design for preheating the air should be avoided, since satisfactory coking-temperatures can be maintained with cold air, if economy of gas is no object. Steam for the operation of the plant should be raised by burning the surplus gas under simple boilers. This simplicity of design permits material economy in cost of construction and maintenance, and also reduces the cost of operation.

The second condition is where there is a market for the gas, perhaps for industrial purposes, but at a low price, and the cost of coal and labor is relatively high. In this case the oven-design should provide for partial heating of the air, with proportionate economy of gas for heating the oven, so that the remaining heat in the waste gases is just about sufficient to furnish the steam required for the plant. This design gives an oven of moderate cost for construction and upkeep; the operation of the boiler-plant is reduced to its lowest terms, with minimum of labor and repairs; and at the same time a very considerable percentage of the total gas produced is available for sale.

In the third case the gas is salable at a good price, so that it is desirable to recover the maximum amount for sale. Under these conditions an oven-design is justifiable which, while more costly in construction, is most economical in heat-consumption. The air is heated in the best form of recuperators or regenerators to 1,000° or 1,100° C., and the waste gases go to the chimney too cool to be of value. Under these conditions coal- and coke-breeze are used for producing the necessary steam. Whether steam is used for the production of power, as well as for the distillation of ammonia, heating, etc., or whether the power is generated by gas-engines, depends upon the relative market value of coal and gas. A proper recognition of the conditions of installation, and the adaptation of the design to them, is an important factor in the most economical installation of a retort oven-plant. It is by no means always the case that the most highly efficient heat-economy within the oven itself is most economical for the plant, and very good results can be obtained in some cases without the most expensive oven-construction. The average of a year's results from two plants which have come within my knowledge, using coal containing

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less than 27 per cent. volatile, showed more than 4,200 ft. of gas sold per ton of coal coked for the entire year. Statements regarding volumes of gas recovered are often misleading, since the quality may vary greatly: so it may be of more interest to say that the heat-units in this gas averaged from the two plants for the entire year well over 2,600,000 B.t.u. per net ton of coal coked, or about 44 per cent. of the heat in the total gas. While these figures can be materially improved by more expensive construction to effect greater heat-economy in the oven, the above results were obtained with the simplest form of apparatus for preheating the air, and at the same time all the steam required for operating the plant, handling and storing coal, distilling ammonia, etc., was produced from the waste heat, assisted by the breeze produced at the plant.

The following tabulated data show in a general way what may be expected from plants designed for the three conditions mentioned above. The figures show the results per ton of coal coked. They represent average conditions and average coals, and, of course, would be modified by special conditions.

Type of Oven.	Fuel Gas. Per Cent.	Surplus Gas. Per Cent.	Steam Produced. Pounds.
No air preheating	70	30	1,050
Partial air preheating	60	40	800
Maximum air preheating	40	60	0

The tendency of modern retort-construction, especially in America, is towards the highest economy of heat, even at the expense of simplicity and economy of construction. The proper heating of a retort oven is not a simple problem. It consists essentially in the distribution of heat from burning gases over the side wall of the oven, which presents an area from 35 to 40 ft. long, and from 8 to 12 ft. high. The temperature over this entire area must be always under accurate control. When operating for maximum output the temperatures must be held at a point not far below the softening-point of the brick; and since the oven-chamber is wider at one end than at the other, to permit easy discharge of the coke, the heats must be modified accordingly, so that the whole charge may be coked in the same time. It is also often desirable to maintain a somewhat lower temperature in the upper part of the oven. The combustion of the gas with air which has

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been preheated to, say, 1,100° C., as it is in the latest regenerator ovens, produces a theoretical flame-temperature of about 2,400° C. The actual temperature produced would be much above the melting-point of the best refractory brick, and while economical operation demands a full utilization of the best temperatures attainable, the walls of the combustion flues must not be injured, nor must there be "hot spots" or "white ends" that will overburn the coke in the adjacent part of the oven-chamber.

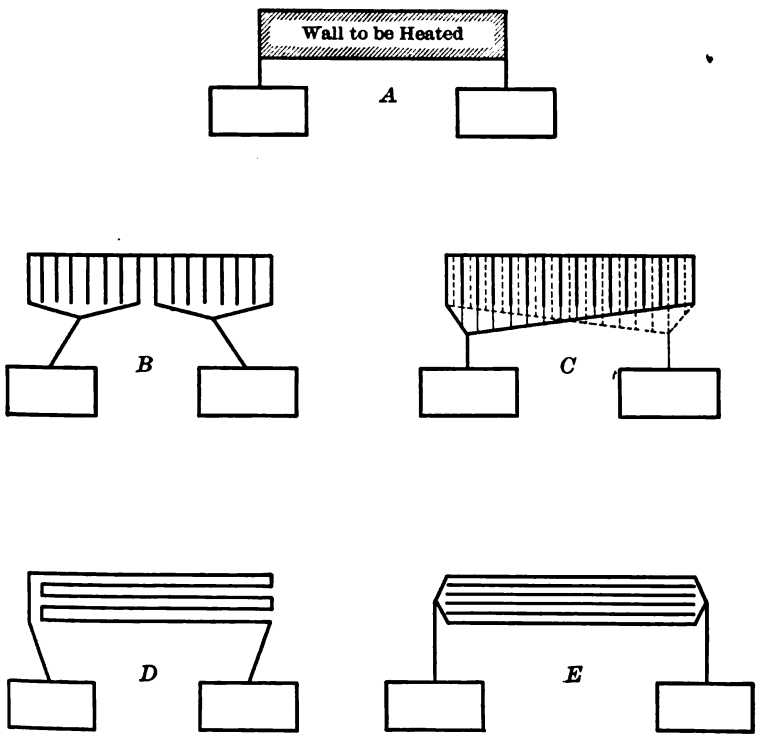

FIG. 1.—APPLICATION OF HEAT IN DIFFERENT TYPES OF COKE-OVENS.

Fig. 1 shows diagrammatically the application of the heat in the different types of ovens. *A* indicates the combustion-flue system of any retort oven with its regenerative chambers below. *B* and *C* show applications of this general arrangement as adopted by oven-systems employing vertical flues. *D* shows the application to the horizontal-flue ovens of the series type. *E* a horizontal type where all the flues are in parallel. *B*, *C*,

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and *D* represent the methods generally adopted in commercial ovens. In the former type, or the vertical-flue oven, the gas and air are mingled and burned in horizontal flues at the bottom of the vertical flues, the burning gases distributing themselves and passing up one-half the vertical flues and down the other half, as indicated in the sketch; or gas and air are led separately to the bottom of each of the vertical flues, where they are mingled by properly arranged nozzles and burned as they pass upward. In some ovens a combination of these methods is adopted. *D* illustrates the horizontal series-flue regenerator oven; and, as I have had more experience with this type, it may be of interest to discuss somewhat in detail the principles of its operation. All of the air required for the combustion is passed through the entire flue-system, first from above downward, and then from below upward, as the regenerator system is reversed. The heating gas is admitted at the end of the flues, usually in four or five places, as may be required. This method seems to give a maximum of simplicity. The flow of gas is automatically reversed in the flues themselves by the change in the current of air. Every flue may be conveniently inspected from end to end by a man walking along in front of the ovens, and the gas-admission at each point is directly under his hand, so that there is no excuse for failure to immediately observe and correct any tendency to unequal heating. The distribution of the heat throughout the flue-system is made remarkably uniform by this process. This is partly due to the comparatively high velocity of the gases sweeping through the flues, and to the fact that the gas is burned either in a large excess of highly-heated air, or else in a mixture of air and products of combustion. The advantages of the presence of products of combustion in the combustion-chamber, where it is desired to obtain a distribution of heat, have been clearly shown in other combustion processes, such as the Doherty system of producer operation, for example.

In beginning the experimental work leading up to the present method of operation, while it was recognized that the presence of a large volume of air, together with products of combustion mixed with the burning gas, was an important factor in distributing the heat with uniformity, yet it was feared that

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passing the entire volume of air through all the flue-system might seriously increase the friction and make the maintenance of sufficient draft a difficult matter, even though the total amount of air admitted could be accurately proportioned so that it would be just sufficient to burn all the gas. Hence, an oven was designed with the flues in parallel, in order to reduce the velocity of the gas-currents, following the principle of the vertical-flue type. It was found, however, that the velocity of the gases passing through the flues did not produce material friction-loss, but that most of the friction arose from eddies in the currents at the ends of the flues; and by a suitable modification of the structure at these points it was easy to keep down the draft required to a reasonable amount, while maintaining the velocity of flow that was so effective in securing uniform heats. This method of handling the gases through the flues in these series-flue ovens has, therefore, two advantages. It is effective in distributing the heat uniformly from one end of the flue to the other, and at the same time the higher velocities effect a more efficient transfer of heat through the flue-wall. The effect of velocity of travel on the transfer of heat through walls has also been shown very clearly in a series of experiments on steam-boilers made by the Engineering Experiment Station of the University of Illinois.[2] It will be remembered that these tests were undertaken to determine the relation of heat-transmission to velocity of circulation in the steam-boiler. They show very clearly the important effect of velocity flow on rate of heat-transmission, and from the deduction therein set forth it would appear that the same conclusions may be drawn in the study of our problem. Having obtained by these means such an effective distribution of the heat, the arrangement of gas-admission to the flue-system makes it easy to control the temperature of the whole system under the severe conditions described above. The result is that in this type of oven the heat is remarkably uniform from end to end, hot or cold spots are practically unknown, and the control of the temperature is much easier than ever before.

The temperatures maintained in the flue-system depend, of course, upon the rate of coking it is desired to maintain. They usually run from 1,000° to 1,250° C., or somewhat higher. The

[2] *Bulletin No.* 40, *University of Illinois* (1909).

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temperature at the point of entrance of the heated air is controlled by the admission of the proper supply of gas, and additional gas is introduced at other points in the system in order to make up for the heat transmitted into the oven-chamber, or to augment the temperatures as may be necessary in preserving the proper relations of heat in the different parts of the system. The periodical reversal of the gas-currents prevents any tendency to inequalities which might otherwise develop.

The perfecting of this system makes the regenerative retort oven quite as simple and easy to operate as the recuperative oven, and at the same time secures the maximum heat-economy. There is only one reversing-valve on each oven-block, by which the currents of air and products of combustion are reversed through the system; and this may be easily operated by a simple mechanical device with time-control. All the other advantages of the older system, of simplicity and reliability of operation, are maintained. This type of oven utilizes the principle of a heavy wall between each two oven-chambers. This wall occupies some room, and therefore more space per unit of production, yet it has several material advantages. It supports the main body of the oven-structure and carries, without difficulty, the weight of the heavy charging-cars, which, when loaded, may weigh more than 20 tons. It permits that part of the oven-structure which is subjected to the highest heats to be readily repaired, or even entirely replaced, without effecting the integrity of the main structure, and also without stopping adjacent ovens. In cases where the coal is hard on the oven bricks, or where a large percentage of water, as in washed coal, causes a severe shock to the red-hot bricks when charged, or after the plant has become old, this is an important feature. Another value of these intermediate walls is that they act as a reservoir of heat. They accumulate heat during the coking process, and when fresh coal is charged they come to the assistance of the burning gas, and help the oven pick up its heat and maintain a coking temperature.

While the earlier by-product oven-plants were simpler in design and equipment, the modern plant is almost entirely mechanically operated, and electrically-driven machinery has taken the place of most of the hand-labor. The following data illus-

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trate the distribution and consumption of power in a plant of a capacity of 1,300 tons of coal per day.

Daily Power Consumption in Kilowatt-Hours for Various Operations.

Lighting	599
Pumps handling ammonia liquor	390
Scrubbers and pumps in by-product recovery-plant	1,283
Coal-charging and coke-pushing	192
Coal-conveyors	393
Coal-unloading	282
Coal-storage	102
Crushing and pulverizing	287
Coke-handling	686
Pumping water	1,800
	6,014

During the development of the by-product oven in the last few years much study has been given to increasing the effectiveness and economy of the apparatus for recovering the by-products. This recovery consists essentially in the cooling of the gas, which causes the condensation of much of the tar-vapor and water-vapor, which latter brings down with it part of the ammonia. After cooling, the gas is scrubbed in contact with water, sometimes by passing it through tall towers through which water is showered, sometimes by bubbling through water in a series of shallow pans, or by other methods; the object in all scrubbing processes being to bring the gas in contact as intimately as possible with wetted surfaces, to take advantage of the strong affinity of water and ammonia. The weak ammonia liquor produced in these processes is distilled by steam, producing either the ammonia liquor of commerce, or, by more refined operations, the purer grades of ammonia. If the ammonia gas from the stills is passed into sulphuric acid, ammonium sulphate, one of the most important artificial fertilizers, is produced.

Much work has been done lately on what are known as direct processes for the recovery of ammonia. In all these processes, the gas is passed directly through a bath of sulphuric acid, thereby turning the ammonia into sulphate. Several such "direct" processes have been worked out, the difference being principally in the method employed for the removal of the tar from the gas, which is then bubbled through the sulphuric acid

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bath; the ammonium sulphate settles out and is removed, and the gas is ready for further treatment. At this point the light oils are removed or the gas is sent direct to the point of consumption. Several methods have been advocated for the utilization of the sulphur in the gas itself as the source of sulphuric acid for making the sulphate. These processes are somewhat complicated, and will not be described here, as they have been fully discussed in the technical journals.

One of the latest applications connecting the manufacture of coke with the other arts is the "gas-oven." This name has been applied to the by-product coke-oven when adapted especially to the manufacture of illuminating-gas. The essential modification is that the gas for heating the ovens is obtained from producers, the ovens being modified to suit this gas, so that the whole of the gas produced from the distillation of the coal in the oven is available for sale as illuminating-gas. This adaptation of the retort oven is attracting considerable attention; several plants are in operation in Europe, and three or four are beginning operation or are in the course of construction in the United States. While producing practically the same gas as other systems employed in coal-gas manufacture, the gas-oven has the advantage of producing a high quality of coke. It also has the advantage in operating-cost, due to the employment of larger units, and carbonizing more coal per unit of labor. These plants are more especially adapted to the larger installations for gas-manufacture.

In discussing the use of by-product coke for metallurgical purposes, the statement is often made that this coke cannot be as good as bee-hive coke, because the quality of the coke is injured in the manufacture in order to obtain better by-products. This is not the fact, as is borne out by the considerable number of by-product oven-plants now supplying metallurgical coke for furnace- and foundry-work. In fact, in a number of localities by-product coke is sold for foundry purposes at a premium over the best bee-hive coke on the market.

One point in which by product oven-operation has improved in recent years is in reliability. In considering the gas, for example, either for gas-engine work or for illuminating purposes, reliability of supply is an absolute essential; and while formerly the supply of gas was uncertain and undependable,

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contracts are now made which insure a permanent and reliable supply that can be depended upon, like any other manufactured product. The sales of oven-gas for illuminating purposes now exceed 40,000,000 ft. per day.

The manufacture of by-product coke is coming to be not only an operation for the manufacture of coke and the saving of such incidental products as may be obtained, but rather an industry where coal is distilled for the purpose of producing several products, such as coke, tar, ammonia, benzol, gas, and perhaps others, each made of the best quality, and each important in maintaining the earnings of the plant. The growth of chemical manufactures in America and the greater demand for the products which supply these industries add permanency and reliability to the market, and at the same time the by-product oven furnishes a reliable supply of raw materials to important industries, which, although in some cases still in embryo here, have attained great proportions in Europe, and doubtless will grow to large importance in this country.

To the student of social economics it appears clear that the general adoption of by-product ovens will exert its influence on commercial conditions, the steady operation of these plants being a factor in maintaining the prices of coke, and hence, indirectly, prices of iron products, thus helping not a little the elimination of those violent fluctuations in iron prices, which have been a fruitful cause of the speculations and industrial depressions which so seriously affect both the employer and the employee. These plants may also have their effect on our industrial life by influencing industries to concentrate in the larger centers in order to take advantage of the raw materials which they furnish, the supply of power which may be obtained from the gas produced, and the opportunities which the plants themselves furnish for the employment of various classes of labor.

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The Manufacture of Coke.

BY F. E. LUCAS, SYDNEY, N. S.

(Cleveland Meeting, October, 1912.)

THIS paper is offered with considerable diffidence, since some of the statements made may not agree with the opinions of other members of the Institute. What I give is the result of some years of experience in the operation and construction of coke-ovens, and observation of plants in America and Europe.

Although my title is "The Manufacture of Coke," I have found it impossible to do justice to the subject without touching on a number of points which, while not strictly involved in the making of coke, yet are of paramount importance from an economic standpoint. There are, indeed, many points of interest, on each of which a separate paper of considerable length might be written, but the scope of my paper will permit me only to touch lightly some of them, in the hope that what I say will promote a full discussion on the whole question. Those of us who are interested in the manufacture of coke are under an obligation to the American Institute of Mining Engineers, which has opened its meetings to the discussion of this subject. I do not believe that there is any question to-day more worthy or more in need of discussion. When geologists and mining-men tell us that, at the present rate of consumption, our coal-fields will be exhausted in a few generations, we should welcome any means that will conserve these deposits as far as possible, and give us the greatest amount of heat or power from every ton mined.

I shall try to show that much can be done in this direction through the by-product coke-oven, and that it should stand second to none as an economic factor in the industrial world.

I shall here deal exclusively with the by-product oven, except for the comparative statement I have prepared, showing the saving to be effected by coking in by-product instead of beehive ovens. It should be borne in mind that this comparison

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is only for one particular coal, and while other coals may give, some better and some poorer results, as far as by-products and yield are concerned, yet it will serve to show the tremendous advantage that by-product ovens have over any other type.

Coke might be called coal with the volatile matter distilled off. And yet coke, as we mean it, and as it must be produced for metallurgical purposes, must be something more than this. It must conform to certain specifications in respect to physical structure and the percentage of impurities present. We have to produce a coke hard enough to carry the weight of the furnace-burden and successfully withstand the abrasion consequent upon the passage of the stock downward through the furnace or cupola; and at the same time, in order to have the best possible quality, it must not be dense. A well-developed cell-structure is essential, I believe, to the best action and lowest consumption of coke in a furnace.

The impurities in coke are principally sulphur and ash, of which the first gives the most trouble.

In America we have in the past been very fortunate in having available large quantities of coals which are naturally of a good coking quality, and require no preparation before charging, except possibly that of crushing to a certain degree of fineness. It is just as true that the exhaustion of these coals is in sight; and it will then be necessary to use other coals of possibly inferior quality. This will necessitate putting the coal through certain processes before charging, in order to maintain the standard of coke.

If the coal is high in sulphur or ash it will have to be washed to remove as much as possible of these impurities. Washing coal consists primarily in keeping the entire mass of coal so agitated in water that the impurities, by reason of their greater specific gravity, may settle to the bottom and be removed. The mechanical devices employed for doing this are many and varied. Some are good, some fairly so, and some bad; but it is not my purpose in this paper to describe or criticize the various types of apparatus. One thing I would, however, emphasize, namely, that any one type of plant does not necessarily work to the best advantage on every coal. Each coal should be carefully studied and tested by all possible means before erecting any plant to prepare it for coking. Different coals have different

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specific gravities, and the ratio between the specific gravity of the coal and that of the impurities also varies greatly in different coals. The manner in which the impurities are associated with the coal also varies greatly, and this in particular will largely affect the size to which the coal should be crushed before washing. We find that some coals will give the best results in washing by crushing to a small size before going to the washer and others give the best results by going to the washer in nut-size. In crushing finely before washing, it is often the case that, where the sulphur is contained in definite strata of pyrites, a large part of the pyrites is crushed to dust or flake form, which is held in suspension in the washer and goes over with the coal. When washing the coarser coal it is (more often than not) found advisable after washing to recrush to the degree of fineness which experience shows will give the best results in the coke for that particular coal.

A further preparation, with which, so far, we have not had to bother much in America, is the compression of the coal into cakes before charging. There are coals which when charged loosely into the oven either will not bond at all, or will not bond sufficiently to make a coke that will stand handling, but which, when crushed very fine and stamped into a cake that can be pushed into the oven, will make a good coke.

It is sometimes claimed that all coal should be stamped before charging, but I cannot agree with that claim, for this reason: a large open cell-structure in the coke, provided the cell-walls are strong enough to carry the burden, gives us a calorific intensity or fierceness of combustion which it is not possible to attain by the use of a dense, close-grained coke; and compressing a coal which gives us under natural conditions such a coke as we desire, only closes in the cell-structure and makes a heavier and denser coke, so that the consumption of coke in a furnace per ton of metal produced would, I believe, be higher, since practically the same bulk, and therefore greater weight, would be used.

However, this does not alter the fact that in many cases compression of the charge is of incalculable benefit to the quality of the coke produced, and as our better coking-coals become exhausted, we shall doubtless find it necessary to resort more largely to both washing and compression, in order to keep

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up the coke-standard. Of course, that will make the coal going to the ovens cost more than at present; but we shall find that other savings can be effected which will more than offset this added cost.

Both washing and compression are almost universally practiced in Europe, but in spite of this we find that the consumption of coke per ton of metal produced is about on a par with the general practice in America, when the quality of the ores and other conditions are considered. Also, we find that the European countries, particularly England, Germany, and Belgium, can, in spite of the fact that the bulk of their coal has to be prepared for coking, and that they are, in general, using much leaner ores, compete very successfully in the world's markets with America, which, as yet, has not been, to any great extent, confronted with the conditions which obtain on the other side. What they have done we can do, and one of the chief reasons why they can compete is because of their avoidance of waste, and their economy in small things. This is most noticeable in the general adoption of by-product ovens and the saving of all possible by-products.

The following description of the process of making coke and recovering by-products will, I think, be applicable to all types of by-product ovens:

The coking-chamber is rectangular in shape, from 30 to 40 ft. long, 6 to 12 ft. high, and 16 to 24 in. wide, taking coal-charges of from 5 to 15 tons per oven. The walls are built to contain an arrangement of flues and chambers in which the gas is burned to keep the oven at the required temperature. Underneath the oven-structure are the regenerators, through which the burnt gases pass after heating the oven-walls, the checker-work in the regenerators absorbing the heat from these gases on their way to the stack. At regular intervals the gas is reversed and passed through a second regenerator, and the air for combustion of the gas is drawn up through the one just heated. By this means air at a temperature of 1,000° to 1,800° F. is supplied for combustion of the gas.

On top of the coking-chamber is a standpipe, or ascension-pipe, connected with the gas-main leading to the condensing-plant.

The coal-charge is either dropped in through holes in the top

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of the oven, or, if compressed, is pushed in through the end door; and then the oven is tightly sealed, to avoid loss of gas, or the possible drawing-in of air, which would lead to a partial combustion of the charge and dilution of the gas. The gas driven off ascends through the standpipe into the collecting-main, and is then drawn to the condensing-plant by means of suitable exhausters.

The recovery of by-products in their first form takes place in the condensing-plant. During the past few years there have been so many new developments in by-product recovery-apparatus that it will be necessary to outline briefly the different processes by means of which the tar and ammonia may be extracted from the gas. The older method was to pass the gas through a series of coolers, bringing it down gradually to the vicinity of from 80° to 90° F., then pass it through the tar-scrubbers, and then through the ammonia-washers, where the gas was brought by various means into intimate contact with water. The water absorbed the ammonia, and this ammonia liquor was then distilled by steam, the ammonia vapor being driven off, and, in turn, absorbed in a bath of sulphuric acid, in which the ammonium sulphate was precipitated, and afterwards extracted and dried, ready for shipment.

The next process was a semi-direct one, in which the gas was cooled sufficiently to allow the tar to be extracted by the ordinary cooling method, and then the gas was heated up again and all passed through the sulphuric acid bath, which absorbed all the ammonia, while the gas passed on to the holder or oven. The sulphate in this case was precipitated and prepared for shipment as in the previous process.

In the semi-direct process a certain amount of weak ammoniacal liquor, consisting of the moisture which was condensed out of the gas in the original cooling, has to be dealt with in stills, the same as in the first process. The quantity of the liquor to be so dealt with will depend on the moisture in the coal charged into the ovens.

A third—the direct—process, consists in taking the gas direct from the ovens without cooling, extracting the tar at this high temperature by specially designed apparatus, and then passing the whole of the gas through the acid bath, as in the previous process. By this method there is no liquor to be dealt with

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and there is also no lime needed in the process, as in the case where liquor has to be distilled and a certain amount of milk of lime has to be added to release the so-called fixed compounds of ammonia in the liquor.

There is still another process, as yet in the experimental stage, in which the gas is cooled and the tar extracted as in the process first described, but the ammonia is recovered in the form of sulphate by utilizing the hydrogen sulphide in the gas instead of sulphuric acid.

A detailed description or discussion of all the different types of ovens and by-product recovery-apparatus would occupy more space than I believe could be granted for this paper.

The coke is discharged from the ovens by means of a ram, and is received either on a quenching-wharf or in one of the many types of quenching-cars or machines. In one type the entire bottom of the oven is dropped, letting the charge fall out into a quenching-car underneath.

The quenching of the coke is important from both the standpoints of moisture-content and of strength. If the coke gets too much water, the cells will suck themselves full as they cool, and there will not be heat enough left to expel this excessive moisture. At the same time, the rapid formation of steam has a tendency to crack the coke and weaken it materially. The ideal quenching of coke can, I think, be shown by immersing a piece in water, just for a moment, immediately after it comes from the ovens, and then leaving it for about 20 min. The water has just chilled the outside and stopped any tendency to combustion there. The inside will die out for want of air, and at the same time there will be sufficient heat to dry out the moisture from the outer cells. It is probably not possible in actual practice to get such results as we would in a small experiment, but the nearer we approach to it, the better the product will be.

The effect of moisture in the coke is one on which opinions may differ. Naturally, if one is buying coke by weight and has to have it hauled a considerable distance, one does not care to pay for the moisture-content or pay freight on it. Even if the moisture is determined and only dry coke paid for, there is still the question of freight, which is an important one when large quantities of coke are being used.

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The effect of moisture in the coke on the action of the furnace or cupola itself, is not so clear to me. In the foundry-cupola, where the ratio of coke to iron melted is so low and the distance between the top and the melting-zone so short, I can readily see how the question of moisture may be very important; but in the blast-furnace we have a very different condition. I do not see how a reasonable amount of moisture in the coke can affect the melting-zone of the furnace or the reducing action of the gas on the ore. There is a temperature at the top of the furnace, where the coke is charged, high enough to thoroughly dry out the moisture before the coke gets far enough down to do any good, and the moisture is carried off with the gas. In fact, I believe there are some furnace-plants where the coke is sprayed with water before charging, in order to keep the top heat down below the danger limit.

Then comes the question of the effect of the moisture on the gas itself. As long as the moisture is not high enough to limit the heats required to be raised on the stoves, I cannot see that it has done any harm.

The effect of longer or shorter coking-hours or higher or lower temperatures on the quality of the coke is something that can only be determined by actual experiment with each different kind of coal. We can make laboratory-tests on a small scale to test the coking-qualities of a coal and its yield of by-products, but I would not care to build a plant to deal with a certain coal and base my calculations entirely on laboratory-tests.

In selecting a design of oven for coking, regard should be had to simplicity as far as possible, accessibility for inspection and repairs, such arrangement of combustion-chambers and flues as will give a uniform temperature in all parts of the oven; and a by-product plant that will turn the gas out free from tar, and with only traces of ammonia, at the same time producing ammonium sulphate of good color: all with due regard to the special conditions presented by a given coal and a given locality.

The advantage of a by-product oven over other types and its value as an economic factor lie not only with the coke, in that it gives a higher yield of coke from a given weight of coal, but also in the by-products and surplus gas.

The ammonium sulphate recovered finds a ready market, and the demand for this product as a fertilizer must always

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increase as the population of the country increases, and the farm- and garden-lands require more fertilization.

The tar also finds a ready market, and this also is bound to increase. The value of tar by-products is rising as the demand increases. It would take a book in itself to describe the many products made from tar, all the way from pitch to drugs and perfumes. But we will consider the main products alone, viz., pitch, creosote oils, and light oils. For road-making, roofing, and briquetting, there will always be a market for the pitch. With the exhaustion of our timber-lands in sight, we must either do something to conserve them or find a substitute. Railroad companies and mine-owners will find it to their advantage to creosote all railroad-ties and pit-timbers as well as bridge- and pier-timber. The lighter oils will be used in internal-combustion engines.

Probably the greatest waste the country has ever seen, or ever will see, has been going on for all the years we have been making coke in bee-hive ovens and burning the gas out in the air. Making coke in by-product ovens, and utilizing the gas by the most economical means, will revolutionize the production of power. It has already done so, most notably in Germany; and we must come to it sooner or later. The sooner we arrive at that point the better for the industries of the country, and the more we shall have done towards conserving our coal-resources.

A modern by-product oven, run at a reasonable capacity, will give 50 per cent. or more of surplus gas from a coal of about 28 per cent. volatile-content. The surplus gas is the gas over and above the quantity needed to keep the oven up to the required temperature. This surplus gas should run from 450 to 500 B.t.u. per cu. ft. The quantity of surplus gas is approximately 5,000 cu. ft.; hence, $5{,}000 \times 450 = 2{,}250{,}000$ B.t.u. per ton of coal carbonized is available for the production of power, = 93,750 B.t.u. per hour. The builders of gas-engines tell us we can get 1 h-p. on a heat-consumption of 11,000 B.t.u. On that basis, we find 8.5 h-p. per hr. from the surplus gas from 1 ton of coal.

The surplus gas can also be used for illuminating purposes. This is done at some plants in this country and at a great many in Germany. By installing two collecting-mains on top of the

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ovens, the rich gas, given off during the earlier hours of the coking time, can be collected in one main, and the leaner gas in the second. By this means gas of 650 to 750 B.t.u., or from 16 to 19 candle-power, can be delivered direct from the ovens without enriching. The lean gas is still of sufficiently high calorific value for heating the ovens. Gas from by-product ovens can be piped for hundreds of miles if necessary. Again, the gas may be used for steam-raising or for heating all manner of furnaces, or, in conjunction with steel-works, can be used in a steel-furnace instead of producer-gas. The recent investigations by Professor Bone have shown how by flameless combustion we can get 95 per cent. of efficiency out of the gas we burn.

In any of the above ways the gas can be used with great economy; but I believe the production of power from gas-engines opens up the largest field.

In the year 1911 there was produced in America approximately 29,338,000 tons of coke, of which approximately 21,448,000 tons was produced in bee-hive ovens. I do not know the figures for the average volatile-content of the coal that went to make this coke, but assuming a fairly low volatile coal of say 24 per cent., to produce 21,000,000 tons of coke in by-product ovens would take about 26,000,000 tons of such coal. Allowing the small amount of 4,000 cu. ft. of surplus gas per ton of coal, and 15 lb. of ammonium sulphate, and 7 gal. of tar, and allowing the surplus gas to furnish only 400 B.t.u. per cu. ft., we find that we would have $26,000,000 \times 4,000 = 104,000,000,000$ cu. ft. of gas per year; this at 400 B.t.u. per cu. ft. $= 41,600,000,000,000$ B.t.u.; reduced to hours $= 4,748,858,447$ B.t.u., at 11,000 B.t.u. per h-p-hr. $= 431,714$ h-p. Or, allowing the value of 10 cents per 1,000 cu. ft. for the gas, we have the sum of $10,400,000. Of ammonium sulphate we have 174,107 gross tons at a value of approximately $60 per ton $= $10,446,420. Of tar we have 182,000,000 gal., worth, at 2 cents per gallon, $3,640,000, Total value of gas, ammonia, and tar, $24,486,420.

I do not doubt that the coal from which the coke was made would have given better results than I have shown here, but even at these conservative figures we can see what a loss there has been.

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The above amount is 7 per cent. on about $350,000,000, a sum which would build by-product ovens enough to carbonize 125,000,000 tons of coal yearly. Besides this loss, there has been the loss of the coal burned in the bee-hive oven. Allowing 64 per cent. as a fair yield for the bee-hive and 78 per cent. for the by-product ovens, there would be a loss exceeding 6,000,000 tons of coal. This at $1 per ton added to the other loss gives us a grand total of over $30,000,000 lost in one year. It seems to me that this is well worth "getting after." Much can be done if we approach this subject in the same manner, and give it the same study and attention as has been given in past years to questions pertaining to mining and metallurgy.

Comparison Between Bee-Hive and By-Product Ovens.

Bee-Hive.

Ordinary type, 12.5 ft. in diameter.

Cost from $700 to $1,200 per oven.

Produces 4 net tons of coke in 48 hr. = 2 net tons in 24 hr.

Yield of coke from coal, 60 per cent.

By-products and surplus gas = none.

By-Product Ovens.

Oven charge, 9 tons.

Coking-time, 24 hr.

(Ovens may be larger or smaller than this, but 9 tons would probably be about the average charge for the modern type of oven.)

Coke produced on 70 per cent. yield = 6.3 tons of coke per oven in 24 hr.

By-Products.

Ammonium sulphate, 22 lb. per net ton of coal = 31 lb. per net ton of coke. Value, 2.25 cents per lb. above cost of manufacture = 70 cents per ton of coke made.

Tar, 8.5 gal. per ton of coal = 10.7 gal. per ton of coke, at 2 cents per gal. = 21 cents per ton of coke.

Surplus gas, 5,000 cu. ft. per ton of coal = 7,143 cu. ft. per ton of coke, at 10 cents per 1,000 cu. ft. = 71 cents per ton of coke.

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Total Value of By-Products as above.

Ammonium sulphate	$0.70
Tar	0.21
Gas	0.71
	$1.62 per ton of coke.

Add to the above the difference between 60 per cent. yield in bee-hive ovens and 70 per cent. in by-product ovens on the same coal. Taking coal at $1.50 per ton :

Coal per ton of coke produced in bee-hive oven	= $2.50
Coal per ton of coke produced in by-product oven	= 2.14
Balance in favor of by-product oven	= $0.36

So that the total saving in coal and by-products equals $1.62 plus $0.36 = $1.98 per ton of coke made, = $12.47 per oven in 24 hr. = $4,551.55 per oven per year. Or for by-products alone, without saving in coal, $3,723 per oven per year.

For a plant of 100 ovens, saving = $455,155 per year.

Cost of 100-oven plant complete, approximately $1,000,000.

A 100-oven plant of above capacity will produce 630 tons of coke per day = 229,950 tons per year, working on 24 hr. coking time.

If benzol is recovered it will further add to the income from by-products.

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By-Product Coke.

BY C. W. ANDREWS, DULUTH, MINN.

(Cleveland Meeting, October, 1912.)

THE various methods of by-product coke-manufacture have been quite thoroughly discussed in a number of papers recently published, and I think it would possibly be of interest and value to discuss the growth and probable future of the by-product coke industry, rather than details of manufacture.

The first by-product coke-plant in the United States, consisting of 12 ovens, was put in operation at Syracuse in January, 1893. This installation was followed by one at Johnstown, Pa., in 1895; one at Dunbar, in 1896, and one at Glassport, in 1897. The plant at Syracuse was primarily intended to supply the owners with ammonia for their alkali-works. At Glassport, for the first time, gas was sold for illuminating purposes, although it was mixed with water-gas to make it marketable. The New England Coke & Gas Co. started the plant in Boston in 1899, which was destined to change the whole trend of the establishment of by-product plants, since it manufactured for the first time illuminating-gas suitable for sale without enrichment. This was accomplished by dividing the gas on the battery into two portions. One portion, consisting of the richer part of the gas, was saved for sale, while the remaining, or leaner, portion was used in the battery. This idea was originated by Dr. Schniewind, of the United Coke & Gas Co. The coke was used for domestic, industrial, and locomotive fuel, as its sulphur-content was too high to make it suitable for use in the foundry. The successful operation of this plant led to the building of additional plants at points where the production of illuminating-gas could be sold to advantage and where there was a fair market for the coke-output.

The following list of illuminating-gas plants in order of construction will give a good idea of the rapid growth of this method of combined gas- and coke-manufacture: Hamilton, Ohio; Detroit, Mich.; Camden, N. J.; Baltimore, Md.; Chester, Pa.; Milwaukee, Wis.; Duluth, Minn.; Geneva, N. Y.; Chicago, Ill.; Indianapolis, Ind. In addition to these there are now under construction plants at Muncie, Ind.; Joliet, Ill., and Waukegan, Ill., all of which will produce illuminating-gas and metallurgical coke.

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During the same period, plants have been built at the following points of pig-iron or steel production: Birmingham district; Benwood, W. Va.; South Sharon, Pa.; Buffalo, N. Y.; Lebanon, Pa.; Steelton, Pa.; Cleveland, Ohio; Joliet, Ill.; Gary, Ind., with plants at Bethlehem, Pa., and Mayville, Wis., under construction.

The above list shows that, with comparatively few exceptions, all the steel-plants and blast-furnaces lying outside of the coking-coal district receive almost their whole coke-supply from the by-product ovens. This is a most decided change from 15 years ago, when Cambria was practically alone in the field.

The question is often asked, why by-product ovens are not being built more freely in the central coking-districts. I think that the following statistics will fully answer this question. In 1907 the total production of coke in the United States was 40,780,000 tons, of which 5,608,000 tons was produced in by-product ovens, and 35,172,000 tons in bee-hive ovens. For the year ending 1910, which was the largest year in the history of the business, the total production was 41,709,000 tons, of which 7,139,000 was by-product, and 34,570,000 was from bee-hive ovens. These data show an actual decrease of 600,000 tons in the output of bee-hive coke as compared with three years previous. The year 1911 made an even poorer showing for the bee-hive, as out of the 35,555,000 tons, 7,848,000 tons was by-product, and 27,707,000 tons was from bee-hive ovens. As a very large proportion of the total bee-hive coke is produced in the central coking-district, it will readily be seen that the present bee-hive ovens are ample to take care of the demand during ordinary years, consequently if the by-product plants were built by the present operators it would result in the loss of the large investment in bee-hive ovens necessary for a corresponding output. In addition, the operators would practically be unable profitably to sell the gas for illuminating purposes, owing to the competition of natural gas. There is no doubt but that the gradual exhaustion of the coal- and gas-fields will eventually lead to the use of by-product ovens throughout the district, since the increase of practically 10 per cent. in yield obtained by the by-product ovens, and a market for illuminating-gas, will eventually become of enough importance to justify the move.

The completion of the plant of the Lehigh Coke Co., near Bethlehem, Pa., and that of the Minnesota Steel Co., at Duluth,

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together with others in prospect, will soon make most of the steel-plants outside of the Pittsburg district self-contained so far as the coke-supply is concerned. There is no doubt in my mind that the by-product oven-gas will take the place of natural gas in a large number of cities in the Central States as soon as the supply of natural gas begins to fail.

As far back as 1898 the use of producer-gas to heat the ovens either wholly or in part was suggested by Dr. Schniewind,[1] and I believe that an experimental installation was actually made at Boston for this purpose. Its use, however, was not extended, owing to various reasons. Within the past two or three years, however, conditions at certain points have become more favorable for this form of gas-production, and several systems have been devised whereby combination ovens are constructed. These ovens are so designed that either producer-gas or lean oven-gas can be used for heating the brick-work. In this way the ovens can be designed to take care of the summer consumption under most favorable conditions, and a producer-gas plant can take care of the increase in the winter consumption. This development will, no doubt, lead to the construction of a large number of comparatively small by-product plants throughout the United States. These plants would ordinarily make sufficient metallurgical coke to supply the needs of the local foundries and sell the remainder of the production for domestic use.

Apparently the final result of by-product oven construction will be the gradual spread of coke-manufacture over practically the whole country instead of in the coking-coal districts. This would result in the conservation of our coking-coal supplies, as not only is a larger yield obtained, but also it is feasible to use coals which are not ordinarily considered suitable for coking in bee-hive ovens. The saving of residuals, such as tar, ammonia, and illuminating-gas, is also of decided commercial importance, since in the United States, before long, large quantities of ammonium sulphate will be used in the manufacture of fertilizers; as well as creosote-oil for wood-preservation and pitch for briquetting certain grades of coal. The domestic sizes of by-product coke are fully equal to anthracite in efficiency, and will form a welcome substitute as the anthracite production diminishes.

[1] *Progressive Age*, vol. xvii., No. 16, p. 366 (Aug. 15, 1899); No. 17, p. 386 (Sept. 1, 1899); vol. xviii., No. 1, p. 9 (Jan. 1, 1900). *Engineering and Mining Journal*, vol. lxvi., No. 15, p. 428 (Oct. 8, 1898); No. 16, p. 458 (Oct. 15, 1898).

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The Manufacture of Coke.

Discussion of the papers of William H. Blauvelt, p. 154; F. E. Lucas, p. 170; and C. W. Andrews, p. 181.

JOSEPH E. THROPP, JR., Indiana Harbor, Ind.:—To what do you attribute the fact that in some localities the by-product coke sells at a premium over the ordinary bee-hive coke for foundry use? If the cell-structure is preferably soft and rather thick in coke for foundry use, as I understand it, do you mean that by-product coke has soft and thick cell-walls?

WILLIAM H. BLAUVELT, Syracuse, N. Y.:—As I say in one of the paragraphs of my paper, the tendency seems to be towards the manufacture of foundry- and furnace-coke under somewhat different conditions. The development of the best cell-structure in coke, of course, has not reached perfection, but we are making progress, and we are inclined to believe that under ordinary conditions the best furnace-coke, that is, coke with the hard cell-wall and the well-developed cell, is produced at fast coking-time and high heats, whereas the best foundry-coke seems to be produced at somewhat lower heats. In one of our plants we are making furnace-coke in 16-hr. coking-time, and foundry-coke in 20-hr. time. Foundry-men want the strong, tough coke.

I think the furnace-men present will agree that the best furnace-practice requires a hard coke, with a well-developed cell-structure. Replying to the question as to the reason for higher prices being realized for by-product foundry-coke than for bee-hive coke, of course, the immediate reason is that it suits the foundry-men better. Perhaps one cause may be that we are able, by the selection of coals, which is not usually practicable at bee-hive plants, to get a quality and analysis best suited to foundry-work, with high carbon-content, and with cell-structure and physical strength especially suited to the foundry-men's ideas.

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JAMES F. KEMP, New York, N. Y.:—Reference has been made to the possible exhaustion of our coal-supply, and its relation to the iron industry. At the Stockholm meeting of the International Geological Congess, two large volumes upon the Iron-Ore Reserves of the World were issued, and it is the purpose at the next session, in Toronto, August, 1913, to publish two companion volumes upon the coal-resources of the world. As in the case of the volumes on Iron-Ore Reserves, the coal volumes will consist of papers prepared by specialists in the several coal-producing countries. They will be at the service of the world at large next August.

E. V. D'INVILLIERS, Philadelphia, Pa.:—On the second page of Mr. Blauvelt's paper it is affirmed that the percentage of oxygen in coal has an effect upon the coking-qualities. What is the minimum percentage of oxygen that is sufficient to show a deficiency? Is any effect had upon the smelting-character of different cokes, and what has been done to overcome that by the mechanical construction of the oven?

MR. BLAUVELT:—The knowledge of the reasons for the coking of coal is in an embryo state, but it is well known that coals which contain a large percentage of oxygen always produce weak and friable coke, if they coke at all.

MR. D'INVILLIERS:—What would be considered a high percentage of oxygen, and is that percentage of oxygen irrespective of other ingredients in the coal?

MR. BLAUVELT:—No, it is not; but it is hard to set a limit. Perhaps 9 or 10 per cent. would begin to be high. Many coals that will coke satisfactorily contain 7 or 8 per cent., but, of course, many run much higher. It is more a question of ratio between oxygen and hydrogen than merely of percentage of oxygen.

Many of the very low volatile coals swell. I have seen some swell so badly in coking that when finally got out of the oven, the coke was gnarled and twisted, and in some cases the coke, in a highly molten state, was pushed out around the edges of the doors. In some cases the force exerted by this swelling-operation is enormous, so that no oven-structure will stand the

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pressure. The best method to counteract the tendency to swell is usually to mix with the coal some other non-swelling coal.

MR. D'INVILLIERS:—I may say further, in regard to my inquiry about the oxygen percentage, that I have nothing whatever to do with the manufacture of coke, but I have a good deal to do with coal-supplies for coking purposes, and I have found that the ultimate analysis of coals, regarded as coking-fuels, would vary from 70 to 80 per cent. in the same coal-field. I was somewhat anxious to know whether the manufacturers of coke, in by-product ovens especially, had a fixed percentage of oxygen, above which they regarded the coal deficient for coking use.

DR. F. SCHNIEWIND, New York, N. Y.:—It is the relation of oxygen to hydrogen which is important. The oxygen freed during the process of distillation will combine with the hydrogen in the proportion of 8 parts of oxygen to 1 of hydrogen. The coking-quality of the coal is greatly a question of the amount of the so-called disposable hydrogen then remaining. The best coking-coals have from 3.5 to 4.5 parts of disposable hydrogen for 100 parts of carbon.

I think the Institute should be congratulated on the presentation of Mr. Blauvelt's very interesting paper, because it goes to the root of the entire by-product coke-oven design. It deserves a discussion which will probably exceed the 5-min. limit.

Mr. Blauvelt says that serial horizontal flues have now been combined for the first time with regenerators. As is well known, the regenerator is the most economical arrangement for recovering "off-heat." The regenerator was originally introduced by G. Hoffmann in the year 1881, and adopted by Dr. C. Otto in his coke-oven design, which was a combination of the Coppée vertical-flue oven with the Hoffmann regenerator. It has been the model for all later coke-oven designs; at least, by far the greater majority of them in Europe are of the vertical-flue regenerative design. I have been for 25 years connected with the building and operation of ovens with vertical flues and regenerators, and I am therefore somewhat skeptical of Mr. Blauvelt's new proposition of a horizontal-flue type with regenerators.

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The general practice of conducting gases in heating processes has been always in the vertical direction. Nobody would think of equipping, for instance, a hot-blast stove with horizontal passes. The natural course of gases giving off heat is the vertical course.

Mr. Blauvelt in his paper then says that in order to obtain uniformity of heating by the combination of horizontal flues and the regenerator, it is necessary to add two things: 1, to carry a ballast of inert, or rather diluent, gases or air; and, 2, to increase greatly the velocity of the gases. This is directly contrary to all experience we have had with the heating of such large bodies as coke-ovens. We believe that slow travel and the effectual exclusion of inert gases are essential to good fuel-economy.

Mr. Blauvelt supports his statements with the experiments which were made at the University of Illinois, in the Experimental Station, and says that in boiler-practice it had been found that high velocity is beneficial. That is undoubtedly true, but heating-conditions in a boiler and in a coke-oven are very different. In boiler-practice the heat-transfer from the gases of combustion to the boiler-tubes is mostly accomplished by convection; in the coke-oven it is practically entirely by radiation. The reason for this is that the temperature-difference in the boiler between heating-gas and boiler-tube, as compared with the temperature-difference in the coke-oven between heating-gas and brick-flue, is so much greater. Where the heat is transmitted by convection it is necessary to remove from the body to be heated the colder film of gases which envelops it. This can be done only by moving the products of combustion at a high degree of velocity. In order to show the difference in the two heating-conditions I have made some rough estimates. In the vertical-flue coke-oven the heated gases passing along 1 sq. ft. of heating-surface carry about 2,000 B.t.u. per hr., while in a boiler this figure is from 3,750 to 4,000 B.t.u. Therefore you see how different the conditions are.

I meant to explain about this question of the transmission of heat by convection. It depends on three factors. In the first place, convection means the transfer of heat from a moving liquid or gas to another body, in this case a solid body, by

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means of direct contact. The frequency of the contact of the molecules in the moving gases with the solid body is therefore the deciding factor in the heat-transmission. In other words, the greater the velocity, as Mr. Blauvelt points out, the greater the transfer of heat, but only in the case of convection. A second factor is the temperature-difference between the gases and the solid body; and the third factor is the specific gravity of the heating-gases. But as we do not transmit heat in the coke-oven by convection, but by radiation derived to a considerable extent from the incandescent carbon particles, we do not need to move the gases so fast. The slower we travel and the less inert material we carry along, the better we find the operating conditions, for it means that we have less differences in the pressures and drafts throughout the entire flue-system within the side-walls.

To sum up this question, I may say that Mr. Blauvelt made the suggestion of a new coke-oven type having a series of horizontal flues combined with regenerators. In order to avoid local superheating he recommended the passing of the flame-gases through the heating-flues at great velocity. But we believe that it is a better plan to sub-divide the places of combustion properly, and to distribute the products of combustion over the vertical-flue system without any unnecessary ballast, and without unnatural velocity.

There are a few other points which I need only to mention shortly. The partition-walls which Mr. Blauvelt mentions are still in use in connection with the horizontal-flue system, but we think that they are rather a detriment. With the modern use of silica brick, for at least 5 years ovens do not need to be touched. It is therefore unnecessary, in our opinion, to provide for such a great brick wall between the heating-flues in order to take care of so little repair-work.

Removing the load of the coal-larry from the top of the oven can be accomplished in a different and less expensive way, viz., by suspending the larry from a bridge, as has been done in a number of plants.

Mr. Blauvelt also mentions that the taper of the oven requires different regulation of the heat at the pusher end, and at the discharge end. This is a very important question. I agree fully with Mr. Blauvelt as to its importance, but I do not

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see, from his oven-design, how this difference of temperature can be obtained, because the gas is fed at both sides, and by the enormous velocity of travel the heat will be carried to the opposite end of the oven, where it is not required. With the vertical-oven system the matter is very easily arranged in two ways, either by changing the time of reversal, giving the narrower side a shorter reversing-period and the wider side a longer reversing-period, or by shifting the partition-wall between the regenerative divisions from the center of the oven towards the wider side.

Mr. Blauvelt also mentions the combination of producers with coke-ovens. This is an important question, but I believe that it is not so much a technical question. These producer connections were tried out 15 years ago and with success. It is rather an economic question. For instance, if a coke-plant is situated in a community where all the gas required by the public can be made in the coke-plant without overcrowding the market for the coke, then the surplus illuminating-gas will generally cost in the holder less than nothing. This situation exists in most of the large cities and manufacturing centers. In such instances producer attachments would be manifestly out of place, because additional gas thus recovered from the coke-ovens by means of producers would cost from 6 to 8 cents per 1,000 cu. ft. It would, therefore, be preferable to erect more coke-ovens instead of producers.

E. W. PARKER, Washington, D. C.:—The gentlemen who have quoted so liberally from the statistical bible in several of the papers presented before this meeting have overlooked one or two items of interest. One of these is that last year the value of the by-products obtained from the manufacture of coke in retort-ovens in the United States was equal practically to the cost at the mines of the coal charged into the ovens. The value of the surplus gas, tar, and ammonia produced at by-product ovens in the United States in 1911 was $10,033,961. The quantity of coal used in the retort-ovens was 10,446,584 short tons, and the value at the mines of the coal in the States from which it was drawn was a little less than $1 per ton. This, of course, does not include freight, which, in the case of retort-

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coke, is borne by the coal, and in the case of bee-hive coke, by the coke. The freight in either case has to be paid, but it can easily be seen that the by-products paid for the coal.

Through the courtesy of the officials of the Pennsylvania Railroad Co. I was able to write into my report on the manufacture of coke in 1911 some interesting estimates regarding the possible utilization of the energy now allowed to go to waste in the Connellsville region of Pennsylvania. The production of coke in the Connellsville and Lower Connellsville districts of Pennsylvania is almost exclusively from bee-hive ovens, of which there are nearly 40,000 in existence. The same quantity of coke if made in retort-ovens would require about one-third the number of ovens. The amount of energy that could be generated from these retort-ovens would be about 180,000 boiler h-p., if no attempt were made to recover the by-products. This boiler h-p. converted into electricity through steam-engines, transmitted from the power-house to sub-stations, and from sub-stations to electric locomotives, would produce more than double the amount of draw-bar h-p. necessary to move every train—freight and passenger—on the Pennsylvania railroad between Pittsburg and Harrisburg. The General Superintendent of Motive Power of the Pennsylvania Railroad Co. had the records of dynamometer, or draw-bar, h-p., calculated for every train moved over that part of the Pennsylvania system during March, 1912. There was a total of 4,107 freight-trains, requiring a total of 32,358 h-p. every minute of the day. There were 4,269 passenger-trains, requiring a total of 9,762 h-p. at any instant, making a total of about 42,000 draw-bar h-p. required at any instant of the day. As the efficiency of generation and transmission is about 52 per cent. of the boiler h-p., it can be seen that about 80,000 h-p. is required at the boiler-plant. This is less than one-half of the total quantity of horse-power wasted every hour of the day, for 365 days in the year, in the Connellsville and Lower Connellsville districts.

Dr. R. Moldenke, Watchung, N. J. (communication to the Secretary *):—There is but little that can be added to the excellent papers by Messrs. Blauvelt and Lucas, since they go into the metallurgical and economic side of the question very

* Presented in oral abstract by Mr. Stoughton.

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fully. Since, however, the requirements of the foundry have been touched upon, a few words along that line may not be amiss.

The question of moisture in coke for foundry purposes resolves itself into that of charging water as fuel. For instance, with a coke containing 18 per cent. of water if taken by weight in charging the cupola—and this is the proper way to proportion charges—there is little wonder if cold iron is obtained instead of the snow-white molten article. And 12, 14, and 18 per cent. of water through exposure to the elements is not an uncommon matter at all in foundry-practice. I have repeatedly seen snow shoveled in with the coke in surprising quantities, as well as coke which had to be sledged apart on account of the ice. Provided that such wet coke does not get into the bed of the cupola—as it will not light up properly and in time—there is no difficulty whatever in using it. With very light cokes the fact that they hold much water is an advantage, as it prevents them from catching fire and wasting too high up above the melting-zone. But the essential point is that a given weight of carbon is provided to melt the required amount of iron. The use of a coke with 3 per cent. of moisture, as compared to the same weight of a coke containing 10 per cent. (entirely apart from the ash-content), may constitute the difference between success and trouble in the foundry. Hence the necessity of attention to uniformity in moisture at the coke-plant, the protection of coke in transit, and the final storage for use in the foundry away from the elements.

Another point is the cell-structure, and incidentally the size of the individual pieces. The foundry-man needs a coke with a cell-wall strong enough to support the shock of dropping pig-iron and heavy scrap some distance upon it. He further needs the percentage of these cells small enough to keep the oxygen busy forming CO_2, and yet not so rapidly that further carbon is taken up to make CO too close to the tuyeres. That is to say, if the reactions go on so that the maximum CO_2, and with it the maximum temperature, is reached at from 18 to 24 in. above the tuyeres under rational blast-volumes, there will be an even distribution of the heat ordinarily, whereas if the maximum temperature is reached nearer the tuyeres, by reason of too high a percentage of cells and ability to burn the carbon, there

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is apt to be an irregular working of the cupola, as gas-channels formed in the bed will permit oxidation of metal more easily, and if the bed thickens up by the use of more coke than is necessary, there will be an excessive reduction in the temperature at the point where the iron is melted.

Here again comes in the size of the lumps of coke. With a lot of breeze, one can readily see how this fine material can work together and thus obstruct the blast, sending it through the more open portions of the fuel. I have seen this happen repeatedly in cupolas of very small diameter, one case particularly where the cupola finally melted on one side only, and as this meant practically double the amount of air blown into the portion which did operate, a furious combustion resulted, with a high melting-loss through oxidation, the slag coming from the tap-hole mixed in with the iron and requiring continual skimming in the ladles before they could be poured. Hence the desirability of having the coke sized to about 4-in. pieces, though anything that remains on a fork with tines 3 in. apart is all right.

Where the coke is porous it is absolutely necessary to make very light metal- and consequent coke-charges, so that the least possible opportunity is given for the formation of CO where the melting is done. The charges of metal being thus close together, heat is absorbed rapidly, and the higher coke-charges do not get incandescent quite so early.

In conclusion, I would say that the by-product method of coking, through the possibility of mixing coals and regulating the temperature, and the ability to market coke unsuited to foundry requirements as produced daily, is in a far better position to satisfy the foundry-man than the bee-hive process can ever hope to be.

Bradley Stoughton, New York, N. Y.:—Probably many here know that recently the use of tar as a fuel in open-hearth steel-furnaces has been taken up by very large steel interests. A member of this Institute several years ago read a paper on that subject and showed that it could be done, but so far as I know it has not been done on a large scale until this year. This of course means more value for the tar produced at the coke-ovens.

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H. H. STOEK, Urbana, Ill.:—The paper by Prof. S. W. Parr, entitled The Coking of Coal at Low Temperatures (*Bulletin 42a* of the University of Illinois Engineering Experiment Station), just issued, is of interest in connection with the present discussion. Professor Parr has demonstrated in his laboratory at the University of Illinois the feasibility of coking in a closed bomb under pressure the so-called non-coking coals of Illinois.

MR. BLAUVELT:—Professor Parr's work brings out some very interesting things which may point the way to work of much commercial value. He makes the point that Illinois coal has too much of the resinous material that produces coke, but he has been able to make satisfactory coke by mixing it with a considerable percentage, I think as much as equal quantities, of coke-breeze. He suggests that the frothy condition of Illinois coke is caused by a large excess of this coking material.

Referring to Dr. Schniewind's discussion of horizontal *versus* vertical flues, of course neither of us will be able to convince the other that he is right. There is one point I did not make clear in my paper, apparently. In the circulation of gas and air with these series regenerator-ovens we do not put in any surplus air as a whole. We start with a surplus of highly-heated air, and then we add gas from time to time during the travel of these gases in the flues. So the first gas burns in a large excess of air, the second gas supplied consumes more of this air, and so on; but when we reach the end of the flue-system we have complete combustion, with no excess of air. So as the gases leave the combustion-flues they are of the same composition as in any other well-constructed coke-oven or combustion-furnace.

Dr. Schniewind asked how we heated the wider end of our oven hotter than the other end. The jets supplying the gas at one end, the current of air soon meets the jet of gas coming in. When the current of gas is flowing downward through the flues, the current deflects that jet downward and supplies that flue with gas. The action is similar when the current is upward, so that does all the reversing of the gas for the whole operation. We have tried in this oven to remove every possible complication that would interfere with obtaining

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the maximum efficiency. The additional heat required by the wider end of the oven is supplied by using somewhat larger burners at that end.

Dr. Schniewind mentioned the middle wall, which is one of the characteristics of the oven I have described. Of course, looked at as a heating-problem, the heating of a coke-oven in daily operation, particularly when coking washed coal, is subject to some rather violent conditions. To heat these walls to more than 1,000° C., and immediately after push out the coke in order to drop in from 10 to 15 tons of coal, perhaps dripping with water, at intervals of from 15 to 20 hr., makes heavy demands on any heating-system, and we find that this wall of brick is an important heat-reservoir in helping the combustion-system to catch up to its work when this charge of fresh coal comes into the oven. The structural advantages, of course, are a matter of opinion. In our modern ovens this middle wall runs from the concrete foundation to the top of the structure. We do not know the life of these walls, but we have never worn them out. It seems to us that the conditions of oven-design call for a construction which is as permanent as possible.

I made mention of the producer-fired oven mainly to point out the state of the art. The application of producer-gas, of course, is not novel, but where it is desired to save all of the gas produced by the carbonization of the coal, producer-gas gives efficient results.

I am sure that Dr. Parker did not mean to convey the impression that, since the value of the by-products may, under some conditions, equal the value of the coal at the mines, any of the furnace-men present can expect to get coke for nothing. There are other items of expense in making coke than the cost of the coal at the mines.

DR. SCHNIEWIND:—Mr. Blauvelt said that he is of the opinion that the ovens are more heated by convection than by radiation, because he cannot see the flame in the flue; but heat-radiation goes on even in a non-luminous flame.

MR. BLAUVELT:—But how can we accept the conclusion that if we have a large body of hot gases flowing through the flues, there is no heating by convection?

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DR. SCHNIEWIND: The heating by convection does not take place to any great extent, because the internal surface of the brick-work of the flues is almost at the temperature of the moving gases. The molecules which are in a state of combustion are generating at the point of combustion a very high temperature, which is immediately transmitted by radiation to the walls. The temperature-difference of the body of moving gases, as compared with the temperature of the brick-work, is so small that convection plays only a minor part in the transmission of heat. I believe that practically the entire heating is done by radiation, and if that is true, the avoiding of any surplus quantity of products of combustion would be right. I believe that that is the common opinion in regard to all metallurgical heating processes also. Every metallurgical heating process tries to exclude the inert or diluent gases as much as possible, and tries to remove the products of combustion as rapidly as possible, as every per cent. of unnecessary oxygen carried through a metallurgical furnace will reduce the efficiency of the furnace.

MR. BLAUVELT:—I was much interested in what Dr. Schniewind said about the transfer of heat by radiation *versus* convection. In our earlier plants our gases burned with a luminous flame, and our conditions were in some respects similar to those he has described in his ovens. But in our modern work, with highly-heated air and gases low in illuminating-power, we do not get a luminous flame. The flues in which the gas is burned appear to be quite clear. Rossetti found in his experiments that the radiation from luminous flame was about three times that of non-luminous flame. I think that those who are familiar with the use of gases in high-temperature furnaces will agree that this is a conservative statement. Little is known regarding the radiation from gases, but it is well known that the quantity of heat radiated from gases is small compared with that radiated from solids. Is it not more probable that under the conditions which we have in these flues heat is transferred from the gases to the wall rather by convection than by radiation? Of course, as the back wall of the flue absorbs heat from the gases, it will radiate an amount of heat, which may be quite considerable, to the wall next the oven-

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chamber, which, of course, is at a lower temperature on account of the travel of the heat through that wall into the coking coal. Moreover, since radiant heat is transferred in accordance with the difference between the fourth powers of the temperatures of the radiant and receiving bodies, while heat is transferred by convection proportionately to the first power of the difference in temperature, it would seem clear that, under the conditions we are considering, radiant heat would be relatively less effective in proportion than if the difference in temperature were greater, as in a boiler, for example. It seems to me, therefore, that the principles which have been developed in the boiler-experiments that I have referred to apply to our problem, and that the rapid flow of gases through the flues does bring in the principle described in the *Bulletin* referred to, and that it partly accounts for the efficient transfer of heat from the gases into the flue-wall which we have observed in practice in our modern oven.

I do not think we need to continue this discussion, as it is getting rather technical. The practical results are what I am endeavoring to present to you. The relations and relative values of radiant and convective heat under these conditions are a difficult subject. But whether the heat is transferred by radiation or by convection, the uniform flow of the gases in this new system carries along and distributes the combustion, so that no matter what temperature may be developed by the burning gases, the heat is carried away from the danger-points, so that there is in practice no tendency to hot spots or other irregularities. These danger-points are immediately removed by the rapid flow of the gases, and the heat throughout the whole system is uniform, and under complete control.

CHARLES H. FULTON, Cleveland, Ohio :—It seems to me that perhaps the use of the word "convection" leads to error in the discussion. If a gas at a high temperature mingles with other gas which is cold, the mixtures assumes a temperature higher than that of the cold gas, and we would say that heating has been done by convection. I do not believe that you can convey heat from a gas to a solid by means of convection, but you can by means of conduction and radiation. I think the fact is well known to metallurgists that a luminous flame will heat

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more readily by radiation than a non-luminous one. In open-hearth furnace-practice it is recognized that if we are to heat readily and rapidly we should have some illuminants in the flame, so that the radiant effect of the flame will be as high as possible. In transferring heat from a gas to a solid which is as poor a conductor as fire-brick, it would seem to me that the length of time of contact, implying a low velocity of the products of combustion, plays an important part. On the other hand, if we convey heat from a gas to a good conductor, such as a boiler-tube, a higher velocity is desirable. I say these few words with the idea that, perhaps, they may throw some light on the discussion.

DR. SCHNIEWIND:—I am familiar with the fact that along the surface of the body to be heated there is a film of gas which is slowly moving, and through that film convection of heat has to take place. As the gentleman just said, the transfer of heat with a small interval of temperature is better accomplished by radiation than by convection.

R. H. SWEETSER, Columbus, Ohio:—How does the oxygen exist in the coal—as oxygen or as a compound of oxygen; and when coal that is stored in stock-piles "absorbs more oxygen," does it take up oxygen as such, or as some compound of oxygen and carbon; and how is the amount of oxygen determined chemically?

DR. SCHNIEWIND:—The oxygen which is taken up by coal is not directly absorbed, but condensed on the surface, and then goes gradually into the coal substance through a slow chemical reaction. We know that coals after being exposed to steam or air or any other kind of gas, accumulate on their surfaces large quantities of gas, which can be expelled by heat. These gases then enter slowly into chemical reaction with the coal. As far as the determination of the oxygen is concerned, it is done by ultimate analysis in the combustion-tube. The question as to the manner in which the oxygen is combined with the carbon nobody can answer. We have been trying to find it out for many years, but no one has explained yet. Notwithstanding the exposure of the coal to the tremendous heat in the coke-

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oven or in a crucible we cannot drive all the oxygen out of the coke; some oxygen will still remain.

MR. BLAUVELT:—The point of the method of heat-transfer is entirely secondary. I offer it in my paper as a suggestion which may account in part for the extremely efficient results which we got by this new method of application of our gas and air. The main point was not that, but rather that the filling of the whole flue-system with a considerable volume enables us to meet, very efficiently, the extremely severe heating-conditions which every by-product oven-chamber demands.

MR. KEMP:—A few years ago we had a very interesting paper from David White, of Washington, upon the percentages of oxygen in coal, and their effect upon calorific values.[1] Mr. White brought out the fact that if we take a long series of elementary analyses of coal, and study the several percentages and then compare their B.t.u. values, it will be evident that the oxygen is inert, and acts in the coal like so much ash, so far as its calorific influence is concerned. It would seem, therefore, as we reflect upon it, that the oxygen of the coal must be all combined in some forms like carbon dioxide, or water, or some similar molecules, and that its combining-power is fully satisfied. That it acts like so much ash I think is now pretty well understood.

F. LOUIS GRAMMER, Leesburg, Va. (communication to the Secretary *):—Mr. Lucas's paper reminds us of our extravagance in coking, and our tardiness in the adoption of by-product ovens, as compared to Germany.

We are, however, so rich in coal—having mined, so far, less than 1 per cent. of our known national resources—that the argument of conservation is not very pressing. It is true that our so-called coking-coal beds are not inexhaustible; but by-product ovens coke coal that bee-hive ovens cannot handle, and so really enlarge our reserves of coking coal.

"Conservation of Resources" is a phrase in great vogue. It is overworked and abused at times; being used by those who desire to be the exclusive custodians of resources or the sole

[1] The Effect of Oxygen in Coal, *Bulletin No.* 382, *U. S. Geological Survey* (1909).

* Received Jan. 27, 1913.

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purveyors of commodities. It does not carry great weight as regards coal.

There are, however, aside from the attractive considerations of profit set forth by Mr. Lucas, several other important reasons why by-product ovens should be introduced more widely.

One is that coal-tar pitch is the keynote to the extension of the coal-briquette industry. The more coal-tar pitch is available, the more briquetting will be carried on; and this will utilize much coal which is either waste or a drug in the market. Heretofore, in several instances coking and steel-making plants have been built, primarily, to market bituminous slack.

Again, a rapidly growing civic pride and an appreciation of the hygienic and psychological disadvantages of smoke are causing restrictions as to the use of soft coal in cities. Several cities require coke to be burnt on locomotives within certain zones, and to be used as exclusive fuel for certain purposes, such as going through tunnels.

Moreover, anthracite is leaving the industrial sphere, and in the domestic circle it will soon be classed with luxuries, such as automobiles, silks, teas, and cigars. Then we may turn to coke as our domestic fuel; especially as we may distribute gas for power, lighting, and heat, directly or indirectly (through the electric current or steam).

Not all by-product coke has been acceptable to the smelters. As suggested by Mr. Lucas, there are interesting fields for investigation here, along the lines of cellular structure. This topic received considerable attention a generation ago in the discussion of charcoal as a fuel. The excellence of charcoal as a metallurgical fuel is not exclusively due to its freedom from sulphur, or its low percentage of ash, or to the alkali in its ash. Even its low temperature of ignition, probably partly due to its softness and porosity, cannot claim to be its other sole virtue. Its porosity in offering numerous surfaces of attack for a given weight makes for rapidity of ignition, small hearth-requirement (notwithstanding its low specific gravity), and calorific intensity.

I have long wished that a thesis might be written by some student of metallurgy on the influence of cell-space on calorific intensity or rapidity of ignition and the influence of ash on the physical strength of the coke. This last should be more easily

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determined; for ash could be added to pure coals. Rumbling, as I suggested about 1896 to a committee appointed to formulate coke-specifications, should be an easy means of comparing coke-strengths.

Some day all our gas may be made in by-product ovens at the mines and piped to our clean, smokeless cities, or piped part of the way, and transformed by combustion so as to send electric energy the rest of the way. It may be questioned whether the direct use of soft coal as a fuel will not be the exception at that future time.

No one should ever build by-product coke-ovens without testing the particular coals in the type of ovens which it is contemplated to use. Too often have enterprises failed, with very disastrous consequences, for lack of this precaution.

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Fuel-Efficiency of the Cupola-Furnace.*

BY JOHN JERMAIN PORTER, STAUNTON, VA.

(Cleveland Meeting, October, 1912.)

THE chief purpose of this paper is to indicate the laws governing the fuel-economy of the cupola, to examine the feasibility of some of the proposals for increasing its fuel-economy, and to show that the conditions limiting fuel-economy here are altogether different from those prevailing in the blast-furnace.

The two methods available for studying the thermal efficiency of the cupola and determining the factors limiting fuel-economy are: 1, by the determination of the total heat-balance; and 2, by the application of the theory of critical temperatures.

The case selected as an example represents a cupola operating under fairly efficient conditions. The data are as follows: Cupola, 60 in. in diameter, 15 ft. high to the charging-door, with a 9-in. lining. Bed-charge, 2,000 lb. of coke and 4,000 lb. of iron. Subsequent charges, 400 lb. of coke and 4,000 lb. of iron. Total number of charges, 20. There was 800 lb. of coke recovered from the drop, hence the total coke burned is 8,800 lb., or 0.11 lb. of coke per lb. of iron. Coke contains 90 per cent. of fixed carbon and 2 per cent. of moisture. 300 lb. of kindling-wood is used in lighting. 80 lb. of limestone (95 per cent. of $CaCO_3$) is used per charge, or 0.02 lb. per lb. of iron. Melting-loss, 4 per cent.; distributed thus: Fe, 3.5; Si, 0.25; Mn, 0.25 per cent. Average analysis of top-gases: CO_2, 15.1; CO, 10.0 per cent. Average temperature of top-gases, 1,600° F. Temperature of air and stock charged, 60° F. Dew-point of air, 50° F. The items entering into the total heat-balance and their calculation are as follows:

1. Heat of combustion of fuel.—Total heat evolved = 14,580 × lb. of carbon burned + 7,200 × lb. of wood burned. Hence B.t.u. per lb. of iron charged =

* Presented also (by title) at the Joint Meeting of the Institute with Section IIIa, Metallurgy and Mining, of the Eighth International Congress of Applied Chemistry, New York, September, 1912.

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$$\frac{8{,}800 \times 0.9 \times 14{,}580 + 300 \times 7{,}200}{80{,}000} = 1{,}470.4.$$

2. Oxidation of iron to FeO.—B.t.u. per lb. of iron charged $= 0.35 \times 2{,}112 = 74.0$.

3. Oxidation of Silicon to SiO_2.—B.t.u. per lb. of iron charged $= 0.0025 \times 12.600 = 31.5$.

4. Oxidation of Manganese to MnO.—B.t.u. per lb. of iron charged $= 0.0025 \times 2{,}975 = 7.4$.

5. Sensible heat in coke.—B.t.u per lb. of iron charged $= 0.11 \times 60 \times 0.16 = 1.1$.

6. Sensible heat in iron.—B.t.u. per lb. of iron charged $= 1 \times 60 \times 0.12 = 7.2$.

7. Sensible heat in limestone.—B.t.u. per lb. of iron charged $= 0.02 \times 60 \times 0.21 = 0.252$.

8. Sensible heat in blast.—From the gas-analysis, 9 lb. of air is used per lb. of carbon burned, hence B.t.u. per lb. of iron charged $= 0.11 \times 0.9 \times 9 \times 60 \times 0.235 = 12.6$.

9. Heat of formation of slag.—This is a matter of some uncertainty, but is of minor importance. The heat of formation of $CaO + SiO_2$ is 278 B.t.u. per lb., and of $FeO + SiO_2$ 121 B.t.u. per lb., and if we assume that the slag consists of equal parts of each, and that 0.06 lb. of slag is made per lb. of iron, the heat of formation of the slag is in B.t.u. per lb. of iron charged $0.06 \times 200 = 12.0$.

1*a. Heat in molten iron.*—B.t.u. per lb. of iron charged $= 0.96 \times 450 = 432.0$.

2*a. Heat in molten slag.*—B.t.u. per lb. of slag $= 1 \times [t \times (0.17 + 0.00004t) +$ latent heat of fusion $+ (t^1 - t) \times 0.35]$, where t = the melting-point of the slag or, say, 2,000° F., and t^1 = the temperature at which it issues from the cupola or, say, 2,250° F. Hence B.t.u. per lb. of iron charged $= 0.06 (2{,}000 \times 0.25 + 160 + 250 \times 0.35) = 44.8$.

3a. Heat to decompose limestone.—B.t.u. per lb. of iron charged $= 0.02 \times 0.95 \times 813 = 15.4$.

4a. Heat to evaporate moisture in coke.—B.t.u. per lb. of iron charged $= 11 \times 0.02 \times 966 = 2.1$.

5a. Heat stored up in lining.—The weight of the lining below the charging-door figures out to approximately 27,400 lb. Estimating its average temperature to be 1,000° F., the B.t.u.

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per lb. of iron charged $= \frac{27{,}400}{80{,}000} \times 1{,}000 \times (0.193 + 0.000043 \times 1{,}000) = 80.8$.

6a. Heat to decompose moisture of blast.—A dew-point of 50° F. corresponds to 0.0075 lb. of water per lb. of moist air. Hence the B.t.u per lb. of iron charged $= 9 \times 0.9 \times 0.11 \times 0.0075 \times 5{,}800 = 38.8$.

7a. Heat sensible in gases.—The weight of the gases per lb. of carbon burned works out as follows: CO_2, 2.200; CO, 0.933; N, 6.910; H, 0.007; total, 10.050 lb. The average specific heat is 0.23 + 0.000023t. Hence the B.t.u. per lb. of iron charged $= 0.11 \times 0.9 \times 10.05 \times 1{,}600 \times 2{,}668 = 424.7$.

8a. Heat potential in gases.—B.t.u. per lb. of iron charged $= 0.11 \times 0.9 \times 0.933 \times 4{,}370 = 403.7$.

9a. Heat lost by radiation plus error and unaccounted for.—This amount is found by difference to be 174.2 B.t.u. per lb. of iron charged.

Summarizing these items, we get the following heat-balance expressed in B.t.u per lb. of iron charged:

Sources of Heat.		Heat Used and Lost.	
1. Combustion of fuel	1,470.4	1a. In molten iron	432.0
2. Oxidation of iron	74.0	2a. In molten slag	44.8
3. Oxidation of silicon	31.5	3a. To decompose limestone	15.4
4. Oxidation of manganese	7.4	4a. To evaporate moisture	2.1
5. Sensible in coke	1.1	5a. To heat up lining	80.8
6. Sensible in iron	7.2	6a. To decompose moisture	38.8
7. Sensible in limestone	0.3	7a. Sensible in gases	424.7
8. Sensible in blast	12.6	8a. Potential in gases	403.7
9. Formation of slag	12.0	9a. Radiation and error	174.2
	1,616.5		1,616.5

It is, of course, true that the limitations to fuel-economy set by the total heat-balance are not necessarily the only limitations. Johnson[1] has pointed out that in the case of the iron blast-furnace, fuel-economy is limited by the heat available and necessary above a "critical temperature," which is, in this case, the free-running temperature of the slag, or about 2,750° F. A similar limitation undoubtedly exists in many other processes, and to test its applicability to the cupola the following calculations have been made:

[1] *Trans.*, xxxvi., 454 (1905).

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In this case the critical temperature may be taken as 2,250° F., and the heat available per lb. of carbon in the fusion-zone and above this temperature works out as follows:

		B.t.u.
Combustion of carbon, 60 per cent. to CO_2, 40 per cent. to CO		10,500.0
Heat brought in by blast at 60° F.		12.6
Heat brought into hearth by carbon at 2,250° F.		905.0
		11,417.6
Less heat to decompose moisture of blast	38.8	
Less heat carried out of hearth by gases	6,371.1	6,409.9
Heat available		5,007.7

The heat necessary per lb. of iron above the critical temperature works out as follows:

		B.t.u.
Heat to fuse iron, latent heat only		65.0
Heat to fuse slag, latent heat only		9.6
Heat stored up in lining, lower part, $\frac{1}{8}$ of total		32.3
Heat lost by radiation, $\frac{1}{4}$ of total		43.5
		150.4
Less heat of oxidation of iron	74.0	
Less heat of oxidation of silicon	31.5	
Less heat of oxidation of manganese	7.4	
Less heat of formation of slag	12.0	124.9
Heat necessary per lb. of iron		25.5

From this we get the somewhat surprising result that there is almost no externally supplied heat necessary in the melting-zone, since the oxidation of the metalloids supplies almost all that is needed to melt the iron and slag. Even in case there were no oxidation here, the theoretical lb. of carbon needed per lb. of iron $= \frac{150.4}{5,007.7} = 0.003$ lb., which is so much below the actual as to show that the heat-requirements of the melting-zone do not impose any limitation to the fuel-economy of the cupola, and that the theory of critical temperature is not adequate to explain its heat-requirements.

Returning to a consideration of the total heat-balance, it is evident that the great source of wasted heat in the cupola is in the gases escaping at the top. If these losses could be eliminated it should be possible to charge some 22 lb. of iron for each lb. of coke, have the gases come off from the top per-

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fectly cold and containing no CO, and the iron satisfactorily melted. Actually this cannot be done, and it remains to be determined why this is so, and how nearly perfection can be reached.

In the cupola there is a deep bed of carbon (coke) which is being replenished from above as fast as it is consumed. Under these conditions, with carbon always in excess, the products of combustion depend upon the temperature and time of contact of the gases with the excess carbon. According to the well-known equilibrium diagram of Boudouard,[2] the tendency is towards the formation of CO at high temperatures and CO_2 at lower temperatures. Now in the cupola there is a zone immediately in front of the tuyeres which is cooled by the inrushing blast of cold air and in which CO_2 is formed, this formation of CO_2 being also aided by the fact that in this space oxygen is supplied faster than the surface of coke present can combine with it. Further in and up in the cupola the temperature is much higher and conditions are such as to favor the reduction of the CO_2 to CO, according to the reaction $CO_2 + C = 2CO$. Time, however, is necessary for this reaction to take place, and since the velocity of the gases is very great and they are in contact with the hot carbon for only an instant, more or less CO_2 invariably passes through unchanged. On the other hand, it is impossible to make the velocity of the gases so great as to prevent entirely the reduction of CO_2 without creating intensely oxidizing conditions inside of the cupola and, hence, destroying its usefulness as a melting-furnace.

The temperature of the top-gases depends on the amount of heat absorbed by the stock in proportion to the total amount generated in the zone of combustion. More heat is generated when carbon is burned to CO_2, and the rapid rate of blowing necessary to the formation of a large percentage of CO_2 increases the velocity of the gases and gives less opportunity for the absorption of heat by the stock.

I have recently investigated the quantitative relationship between rate of blowing and temperature and composition of top-gases in the operation of commercial cupolas, and the results of my experiments, as far as they go, are summarized in Figs. 1 to 5.

[2] *Annales de Chimie et de Physique*, Seventh Series, vol. xxiv., pp. 5 to 85 (1901).

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Fig. 1 shows some typical top-gas temperature records during the progress of the heat. Each line represents a separate cupola. It will be noted that there is great irregularity in the temperature-line up to the time charging ceases, this being due to the intermittent cooling by freshly-charged material, the influence of temporary holes in the stock, etc. After charging ceases, however, it is invariably the case that the temperature rises rapidly to a maximum approximately equal to the melting-point of cast-iron. This, of course, is due to the shortening column of stock and the absence of fresh additions of cold stock.

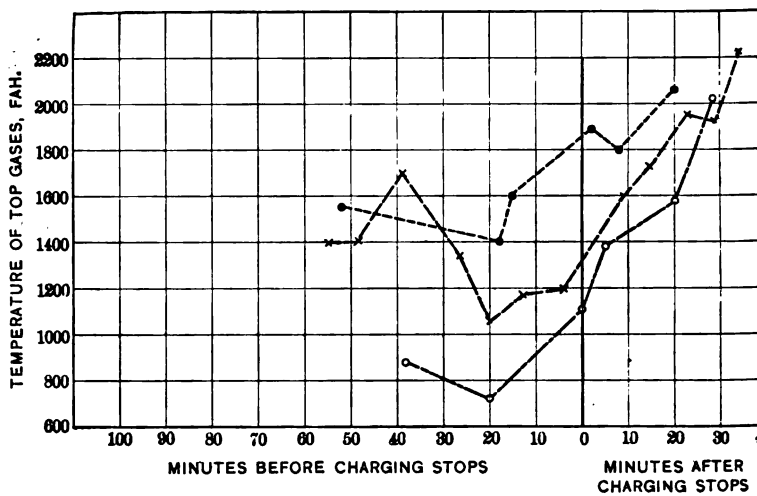

FIG. 1.—TIME-TEMPERATURE CURVES OF TOP-GASES.

Fig. 2 shows some typical top-gas composition records during the progress of the heat. In this case there is even greater irregularity, since the disturbing influences are greater. There is no well-defined tendency towards either increase or decrease in CO, the effect of the higher temperature after charging stops being more or less neutralized by the shortening column of fuel.

Fig. 3 shows the approximate relation between rate of blowing and average composition of top-gas up to the time charging stops. Each of the six points from which this curve was plotted represents a separate cupola, and owing to the disturbing effect of different heights and different brands of coke it is natural

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that there should be some irregularity. The results as a whole, however, are fairly consistent.

Fig. 4 shows the approximate relation between rate of blowing and average temperature of top gases up to the time charging stops. Each of the points represents a separate cupola, and as

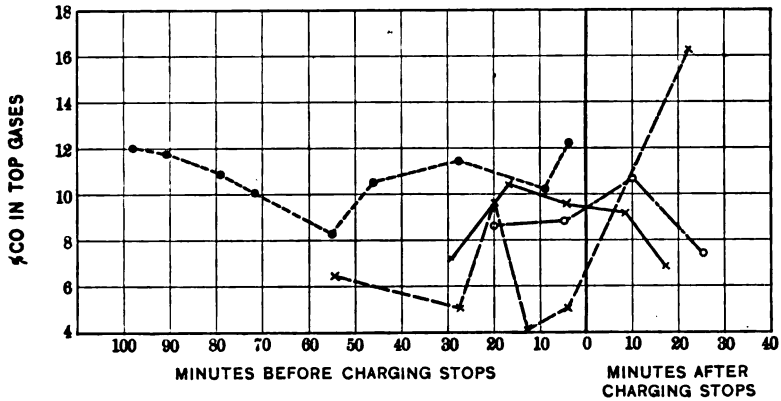

FIG. 2.—CURVES OF TIME AND GAS-COMPOSITION.

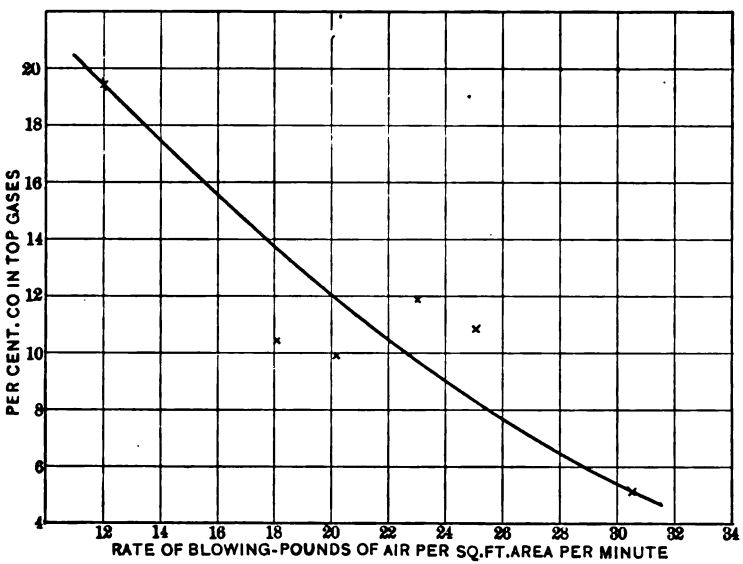

FIG. 3.—RELATION BETWEEN RATE OF BLOWING AND GAS-COMPOSITION.

the heights of the cupolas and other factors varied considerably, it is natural that there should be some irregularity. On the whole, however, there is a marked increase in temperature with increased rate of blowing.

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Fig. 5 shows graphically the approximate relationship of heat-loss, sensible and potential, to rate of blowing. The curves are calculated from the preceding data, which, however, are too scanty to give very reliable results. Incidentally it may be stated that from 25 to 30 lb. of air per sq. ft. of cupola-area per min. has been found by experience to be the most advantageous rate of blowing, all things considered.

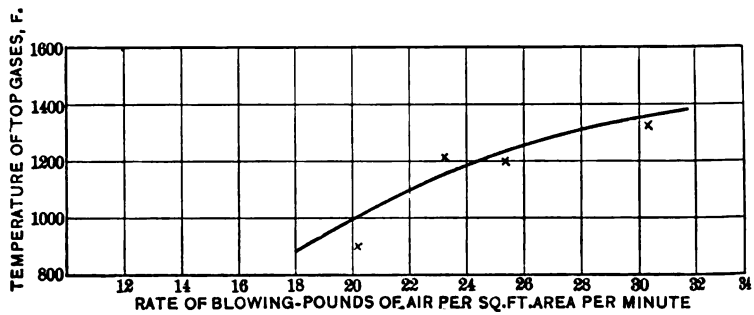

FIG. 4.—RELATION BETWEEN RATE OF BLOWING AND TEMPERATURE OF GASES.

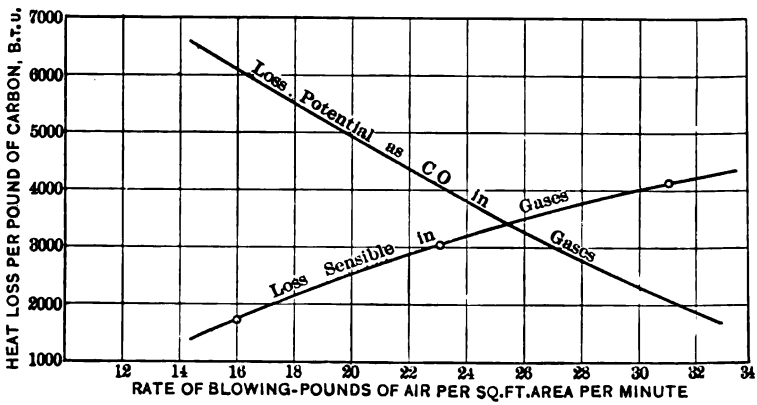

FIG. 5.—RELATION BETWEEN RATE OF BLOWING AND HEAT-LOSSES IN THE CUPOLA.

Within recent years there have been five principal proposals made with a view to fuel-economy in cupola-operation. These are, increased height of cupola, a second row of tuyeres, heated blast, dry blast, and the Balliot heat-recovery cupola.

The first of these, increased height of cupola, has been shown to rest upon a sound theoretical basis, while the results of practice indicate that it accomplishes the results desired. The

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quantitative relation between increase in height and fuel-saving does not, however, seem to have been determined.

A second row of tuyeres is supposed to effect the more complete combustion of the carbon by the addition of oxygen at a point where it can burn the CO. Theoretically this appears to be an excellent idea, but actually it usually fails to accomplish the result desired, *i. e.*, to decrease materially the proportion of CO, although it may produce other desirable results.

Since CO is not formed directly at the tuyeres, but only at some distance above or in front of them, it is evident that the second row of tuyeres must be placed a considerable distance above the lower ones in order to reach the CO. Now, if this is done, the result is to produce another zone of combustion and a practical duplication of conditions at the lower tuyeres. Where the addition of a second row has proved advantageous the benefit can usually be traced to the enlargement of the total tuyere-area and the consequent greater volume of air admitted to the cupola.

In the blast-furnace the factor limiting fuel-economy is the heat available above a critical temperature of about 2,750° F., which is only a small percentage of the total heat. The heat in the blast, while only a small item in the total heat-balance, is a direct addition to the available heat, which it may increase 100 per cent. or more. In the cupola, on the other hand, the limiting factor is the total heat present, of which the hot blast can only be a very small percentage. For this reason the saving due to hot blast in blast-furnace practice cannot be taken as a criterion of what to expect in cupola practice.

In the case of the cupola previously considered, heating the blast to 1,000° F. would mean an addition of 2,204 B.t.u. per lb. of carbon burned, which theoretically should reduce the carbon-consumption to 0.083 lb. per lb. of iron, or a saving of 16 per cent. Hence the saving in cupola practice is approximately only one-third of that found in the blast-furnace.

The use of an air-blast artificially dried by refrigeration has proved of very great advantage in blast-furnace practice from the standpoint of fuel-economy and increased output. Reasoning by analogy, some metallurgists have proposed to apply the process to the cupola, predicting equally good results here. The experiment has never been actually tried, but it can

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readily be shown that the saving would be very much smaller in the case of the cupola.

The value of dry blast in the blast-furnace depends upon the great increase in heat available above the critical temperature which it creates. As already explained, this is not a limiting factor in the cupola, and the only fuel-saving to be expected here is given by the ratio of the heat of decomposition of the moisture in the blast to the total heat generated per lb. of carbon, or in the case used as an example $\frac{38.8}{10,500} = 0.38$ lb. of carbon, or about 3.8 per cent.

In the Balliot cupola[3] the main feature is the return of a portion of the gases of combustion to the wind-box, and thence into the zone of combustion again, the idea being similar to the Eldred process of flame tempering as applied to lime-burning. The result of this procedure is to dilute the air entering the cupola with inert gases, to enlarge greatly the zone of combustion, to lower the temperature of this zone, and probably to make it more uniform, and finally to increase greatly the volume of gases passing up through the cupola stack, though not to increase the total amount passing out of the cupola.

It is claimed that the net result of this procedure is to enable melting to be done with less fuel and less oxidation of the iron. Our theories do not seem adequate to test these claims satisfactorily, but they at least indicate their possibility. The effect of introducing inert gases with the air is exactly opposite to the effect of hot and dry blast as regards size and temperature of the combustion-zone. If the cupola were similar to the blast-furnace as regards dependence upon heat available above a critical temperature, this process would necessarily cause a great loss in fuel-economy. As we have seen, however, this is not the case.

On the other hand, it is difficult to show theoretically where there is any gain in fuel-efficiency. If there is such a gain, it is probably to be attributed to more uniform working caused by the larger zone of combustion. It seems that the claim of less oxidation due to lower melting-temperature and more uniform distribution of the heat is at least plausible.

[3] Chemical Reactions in Foundry Cupola Practice, by J. De Clercy. *Transactions of the American Foundrymen's Association*, vol. xvi., p. 7 (1908). Distribution of Temperatures in the Cupola, by J. De Clercy. *Ibid.*, p. 103. Discussion on Cupola Practice. *Ibid.*, p. 154.

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Melting Iron in the Cupola-Furnace.

BY DR. R. MOLDENKE, WATCHUNG, N. J.

(Cleveland Meeting, October, 1912.)

UNLIKE the furnaces employed in the reduction of ores to mattes and metals, the foundry-cupola has only melting to do. This looks simple enough; and its development has progressed through centuries by cut-and-try methods; but there is still much to be learned concerning the causes which underlie the production of defective castings, even from first-rate materials.

There are other differences between the smelter and the cupola. The former is usually, the latter rarely, run continuously. In the smelter there is some chance for experiment, since rich slags can be treated again; while in the cupola the metal must be good for casting from the first tap to the drop of the bottom; otherwise, heavy damage will result. In the smelter, the object sought is the production of metal or matte with a minimum loss in the slag; whereas, in the foundry, a variety of castings of different compositions must often be run from the same heat, one after the other.

Experience with the cupola has shown that the melting should take place, throughout the heat, in one zone, the zone of maximum temperature, which is clearly marked on the cupola-lining by the severe cutting action of the slag and iron oxide. The design and operation of the cupola must conform to this condition. This conformity having been secured, other considerations resolve themselves into those details of construction which will involve least trouble and cost of repairs.

The very high temperatures required to melt iron and steel in the cupola, leaving the metal sufficiently superheated for casting, render the use of water-jackets inadvisable. It is always possible to patch up the lining at the melting-zone between daily heats; and the rest of the lining is seldom affected enough by abrasion and chemical action to require replacement oftener than once in nine months.

The cupola is practically a shell of steel, lined with the proper refractory material, and provided with a set of tuyeres for the admission of air to burn the fuel and melt the metallic

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charge. It should have the same diameter throughout, since there is no reduction of ore and consequent change in bulk of material. The tuyeres should give an almost continuous opening, so as to keep the zone of maximum temperature (as indicated on the lining above them) practically horizontal. They are placed at such a distance above the bottom as to allow for the slag below them, and still leave sufficient room in the hearth for the required quantity of molten iron, without danger that it will run from the slag-spout, or through the tuyeres into the wind-box.

In cupola-melting it is necessary to burn the fuel as completely as possible, in order to attain the maximum available temperature. That is to say, the carbon of the coke or anthracite—whichever happens to be used—must be converted to carbon dioxide with as little subsequent change to carbon monoxide (by taking up extra carbon from the incandescent fuel) as may be possible. If this desirable condition is effected at the point where the metal charge has been placed, melting under the most favorable conditions ensues. The molten iron is superheated properly, and the chances of oxidation in melting are reduced to a minimum.

It follows that in order to get the carbon of the fuel converted into the maximum amount of carbon dioxide, not only must the bed stop at the point where this is secured, but the amount of air blown into the cupola and the diameter of the cupola must stand in some fixed relation. It is well, therefore, to study the *rationale* of the process of combustion which takes place in a cupola. Let us suppose that the coke-bed of a given cupola has been charged and well burned through, the first metal charge has been put on, and then the succeeding intermediate charge of coke, which is intended only to replace that portion of the bed burned away in melting the first metal charge. The second metal charge is then put on, then coke, then metal, and so on. It is desired to start up melting. The blast is put on, and this is what happens: As the air enters through the tuyeres every molecule of oxygen that touches the incandescent fuel picks up enough carbon to make a molecule of carbon dioxide, which travels upward through the incandescent bed. Now some of this carbon dioxide must naturally be changed to carbon monoxide by contact with fresh incandescent coke, but there is much free oxygen present, as well as the inert nitrogen,

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which serves to retard the change from CO_2 to CO. As a matter of fact, the maximum proportion of CO_2, under ordinary conditions, is reached at 18 to 24 in. above the tuyeres, the bed being sufficiently thick to allow this. If more incandescent fuel is above this (as would be the case in producer-gas practice), then the change from CO_2 to CO is rapid, and the poor melting-results are quickly noticeable.

It will be seen that it is quite easy for some free oxygen to penetrate a considerable distance up into the charges before being used up in the combustion. In fact, tests on this point by the U. S. Bureau of Mines have shown conclusively that there is no place in the cupola absolutely free from uncombined oxygen. It was further found that much air escaped practically unchanged along the lining, and hence the danger in charging all the heavy iron along the lining, where it is in just the right place to become oxidized as it gets into the melting-zone.

Another interesting point proved by the investigations of the Bureau of Mines is, that there is a central cone in the fuel-bed of the cupola above the tuyeres in which there is a formation of CO only, showing that no complete combustion goes on there, which indicates that as the air is blown into the cupola it curves upward, and some of it does not reach the center directly opposite the tuyeres. The smaller the amount of air going into the cupola, the higher this cone will be; and if it should extend beyond the original height of the bed (after melting for some time) by accumulations from the subsequent coke charges, there will be a diversion of the metal charges outward from the center as they descend, and the melting done in the cupola will be considerably less than the normal amount, besides going on in a very uneven manner. On the other hand, the more air is blown into the cupola, the shorter becomes this cone, and it disappears altogether when the air is forced straight through the bed. This is the ideal condition; and any further forcing of the air by using larger quantities will unduly increase the melting-capacity of the cupola, compel the raising of the coke-bed, and be likely to start channels in it which will conduct all the air through a portion of the bed only, consequently oxidizing much iron, and proving generally unsatisfactory.

The above somewhat involved explanation shows why the amount of air blown into a cupola should bear a certain rela-

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tion to the diameter of the furnace. Practice is the best guide to this. For instance, the ordinary cupola, with a diameter of 54 in. inside the lining, under the best conditions of practice has been found to give 10 tons an hour. Since it takes a little less than 30,000 cu. ft. of air to melt a ton of iron, it will be necessary to provide 300,000 cu. ft. of air per hour for that cupola. It is possible to get good results with less air; but then the melting-rate drops and this is bad foundry-economy. On the other hand, it is possible to get 11 or even 12 tons an hour from the cupola, but this means blowing in the corresponding amount of air, with consequent chances for bad working.

There is therefore a rational rate of melting for each diameter of cupola; and this is given by the catalogues of all makers, as taken from experience. If a given cupola does not perform in accordance with this rate and the amount of air blown in has been found to be the correct one—allowing 30,000 cu. ft. per ton to be melted—then the cause of the trouble must be looked for elsewhere, as will be shown below.

Taking up next the question of cupola-temperature: as the blast goes through the fuel-bed the gases become hotter and hotter up to the point of maximum proportion of CO_2, and this will be about 3,900° F. At the zone of this temperature, which is from 18 to 24 in. above the tuyeres, as previously stated, the hot gases should find the charge of metal to melt. Should too much fuel have been charged originally, the conversion to CO takes place, with consequent reduction of the temperature. It will therefore be seen that from the instant the air touches the fuel there is a rapid increase in temperature upward until the maximum is reached, and then a decrease. Experience shows that melting iron is possible for a distance of about 1 ft. below this place of maximum temperature, and perhaps 2 ft. above it, if the bed has been allowed to be so low or so high through improper charging. This effect can be readily understood when it is remembered that the melting-point of white iron runs as low as 2,000°F., while that of gray iron is several hundred degrees higher. The iron melts; but in the case of too low a bed it will be insufficiently superheated, besides having been exposed to free oxygen, with all the troubles this brings about. In the case of too high a bed, the metal will not have been oxidized, but will be so cold that even dropping

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through a hotter portion of the bed will not give it the proper temperature for casting.

To find the proper height of bed—and this height is a function, not of the weight of the fuel, but of the amount of travel through it which the air has to perform until the maximum CO_2 has resulted, and with it the maximum temperature—experience again has taught us. It has been observed that when conditions in the cupola are just right for "blast to go on," which is to say, when the bed is well burned through, and the cupola charged to near the top, with the metal heating up satisfactorily, it should take between 8 and 10 min. from the commencement of blowing until enough iron runs from the spout to necessitate closing up the tap-hole. The melting iron will be observed dropping by the peep-holes in the wind-box in about 5 min. after the blast is on, but it takes a little longer before there is enough metal to begin running out. If, under these conditions, the metal comes in less than 8 min., the bed is too thin, and should be increased by charging a little more fuel between the metal charges in order to bring it up properly. If longer than 10 min., the bed is too thick and should be correspondingly reduced.

Considering the actual function of the bed, in view of what has been said so far, we find that every portion of it below the tuyeres is simply so much filling intended to give storage space between the lumps of incandescent coke for the molten metal. Above the tuyeres we find the chemical reactions of combustion going on, which result in maximum temperature-conditions at a given zone, in which melting should be done. Only the upper few inches of this bed is at the maximum temperature, and below this the bed is cooler and contains the dreaded free oxygen. Hence the metal charge should be so proportioned that it is melted down completely by the time about 4 in. of the bed has been consumed. The correctness of this statement can be observed by examining the lining of a cupola. In a well-regulated foundry the scoring of the lining is confined to a belt of from 4 to 6 in. Where this ideal condition does not obtain the belt of affected cupola-lining may be 3 ft. high. The latter condition shows a shifting of the melting-zone up and down, according as the bed had been allowed to burn away before the succeeding charges of coke came down to replace it, or the

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bed was allowed to build up by giving too much coke between the charges of metal.

The inferences that must be made from the above are: (1) that the smaller the charge of metal and the oftener repeated, the less the shifting of the melting-zone up and down in the cupola; (2) that the almost universal rule in the United States of making the first metal charge twice as heavy as the succeeding ones is a serious mistake. Where this is done, it is patent that double the amount of coke must have been burned away from the top of the bed, only half of this being replaced by the intermediate charge of coke, and from the second charge of metal on, the melting is done at a lower zone in the cupola. It is difficult to instill this point into the minds of cupola-men and even chemists. They see a big pile of coke go into the cupola for the bed, and of necessity hold that a big lot of iron should go on it, forgetting that only the uppermost portion of the bed does the work whether the bed is big or little. European practice is more rational in this respect, for not only are cupolas of small diameter used, with consequent effective penetration of blast, but the charges are very small and there is no large first charge. Like many other things, we "over-improved" the melting process when we began doing big things here.

Once the proper thickness of the bed has been found for a given coke, the replacement of what is burned away in melting each charge is a definite function of its composition and somewhat of its cellular structure. Every time the brand of coke is changed it is necessary to try out the melting-time for "first iron." In fact, in well-regulated shops this is done every day—unless the shop custom is to keep the breast closed during the "burning through" and "blast on," when it is sufficient to make the above described time-trial once a week. It is also necessary that the fuel for the bed should be perfectly dry, though this is not essential for the intermediate charges, provided due allowance is made for the water in weighing up, so as to secure the right proportion of carbon.

It is therefore well to use another "experience" figure for the weight of the intermediate fuel. This should be one-tenth the weight of the metal to be charged above. A good high fixed-carbon coke can melt more than ten times its weight, or the ratio is, say, 11 lb. of metal to 1 lb. of coke, while a high-ash

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coke works the other way. Hence we have got to the point where in trying out the melting-conditions required for a cupola, the first iron is wanted at, say, 8 min., the first charge of metal as small as all the others, and the charges of coke between these metal charges one tenth in weight. It now remains to fix the size of the charges of metal.

Since only about 4 in. of the coke-bed should be burned away in melting the metal charge above it before the next coke charge comes down, it stands to reason that the proper size of the metal charge should be that requiring these 4 in. of coke in melting, or 10 times the weight of 4 in. thickness of coke in the cupola. The best way to ascertain this weight is to lay a ring of cupola-blocks or fire-brick on the cupola charging-floor, of the diameter of the cupola and 4 in. high. Fill this space with the coke in question and weigh it. Ten times the weight is the amount of metal to be charged. Really the ideal charging method is to take the proper proportions of pig, scrap, and coke, and mix them together as they are charged; thus, in melting, a lump of coke replaces one just being burned away, and there is no shifting of the melting-zone. It is an expensive way, however, and not likely to be carried out properly.

Inasmuch as during the course of a melt the cupola becomes hotter, and the upper charges well heated, less coke is required between the charges; and here the experience of the foundry-man comes into play. The easiest method of determining this question is to observe the rate in melting. That is to say, if a given cupola melts 10 tons an hour for the first hour, and only 9 tons the second hour, it will have been because of excessive coke (other conditions being right), and a gradual and slight reduction of the intermediate charges of coke is in order, until the melting-rate is restored and even accelerated a little. Continued observation quickly overcomes this difficulty.

It might be said here very emphatically that the current practice of giving a bonus in the foundry for saving coke is radically wrong. There is only one right proportion of coke to use. If more is used the melter should be replaced. If less, the manager is in fault in allowing it. If the foregoing deductions have been followed closely, it will be realized how easily a cupola can be charged and run. In the case of hundreds of cupolas coming under my observation for correction of troubles, the application of the reasoning given above, and the corres-

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ponding changes in practice, have almost immediately overcome the difficulty.

When, in melting, iron gets into portions of the cupola where free oxygen is present, it is affected thereby, and the result is a higher freezing-point. The metal loses its "life" and cannot safely be held for any time. Moreover, it contains gases, which come out at the moment of set, with the result that the castings show evidences of pin-holes, heavy shrinkages, and even cracks from loss of power to accommodate themselves to internal strains while the metal is setting. And still worse, the pin-holes in question often do not appear until the skin of the castings is removed by machining. This is because when the mold is poured, the metal immediately in contact with the sand sets first, and in doing so passes its contained gas through it. Once this skin has formed, further gases attempting to get out are shut off, rise to the top, and will remain just under the skin. Metal of this kind always shows defects when the cope portion is planed off. The bottom of the casting as poured may be all right.

Since this condition is entirely due to the oxidation of the metal, it is important that the charging of the cupola be done very evenly and regularly, and that the pigs be not charged around the lining, with the scrap in the center. In the latter case, not only will the bulk of the metal be exposed to the gases rich in free oxygen which pass up along the walls, but there will be practically 16 lb. or more of metal to be melted by the pound of coke underneath along the outside of the charges, and say 5 lb. of metal only for the pound of coke underneath in the center. The result can only be trouble, especially where small castings are made; for the comparatively cold iron from the outer portion of the cupola, while mixing with the very hot interior portion, may give a mixture hot enough to pour all right, but that 16-to-1 metal has been damaged by oxidation, and this leaves its effect on the mixture.

The subject could be extended still further by giving many and varied examples from actual experience, but enough has been said to indicate that the cupola is by no means the simple contrivance that it is supposed to be, and that there are refinements in practice which take into account peculiarities in fuel, metal, and even the air used for combustion.

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In summing up the subject, the following conditions should be observed:

1. That the proper amount of air get into the cupola for its capacity.

2. That the bed-coke be dry and well lighted before charging.

3. That the bed be of proper height to give "first iron" in from 8 to 10 min.

4. That the metal charges be of equal weight.

5. That the metal charges be no larger than will be just covered by the coke required to melt them.

6. That the coke charges be so adjusted to the metal charges that, throughout the heat, the melting-zone remains stationary and at the right point.

7. That the blast-volume (not pressure) never change throughout the heat, since any variation will immediately change the position of the melting-zone.

8. That the charges be evenly distributed: first, the pig-iron over the entire bed; then scrap also over the entire bed; then coke. If steel is used, put it on before the pig. Never use thin scrap-steel.

9. That only one row of tuyeres be used, and that these be of sufficient size. If a second row be available, open only a very few of them, so as not to disturb the position of the melting-zone, while giving extra air to burn the CO that may form too low in the furnace.

10. That the melting-rate be watched and the intermediate coke-charges adjusted accordingly.

11. That the charging of very large pieces of metal be avoided, since these very often deflect the gas-currents and bring about an uneven burning of the fuel.

Heavy coke, with small percentages of cell-space, can stand large charges, anthracite coal being the extreme in this connection. Light cokes, with large percentages of cell-space, must have very small charges to get best results.

Where the above suggestions have been tried, the result has invariably been snow-white molten iron from the spout of the cupola, perfect mixing of the iron charged, and sound castings. It is to be understood that the present paper covers only the metallurgy of the cupola melting process, and not the mechanical considerations looking towards extreme economy in operation.

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The Methods of the United States Steel Corporation for the Commercial Sampling and Analysis of Pig-Iron.*

BY J. M. CAMP,† PITTSBURG, PA.

(Cleveland Meeting, October, 1912.)

I. PREFACE.

IN conformity with the design of the officials of the United States Steel Corporation for the standardization of the methods employed in the sampling and analysis of all materials encountered in their various lines of manufacture, the Chemists' Committee presents this compilation of standard methods for the sampling and subsequent analysis of molten pig-iron.

In selecting the methods, the Committee employed the same line of procedure as in former cases, viz., the careful consideration of all the methods employed in each laboratory of the Corporation, evolving therefrom the several methods herein described, the immediate adoption of which is desired.

The services of Messrs. W. B. N. Hawk, William Brady, and C. H. Rich, a sub-committee appointed for the preparation of the present paper, are gratefully acknowledged.

II. INTRODUCTION.

A quite obvious cause of nonconformity of results of comparative analyses in the various laboratories of the Corporation has been an apparent lack of uniformity of method in the sampling and analysis of molten iron.

This condition is particularly apparent with regard to the determination of the sulphur, numerous discrepancies in which would appear to indicate the absence of, and the necessity for, some uniform method of procedure. The estimation of the

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sulphur by the gravimetric method and the determination of the remaining constituent elements of the iron, are susceptible of a most satisfactory degree of accuracy. The successful issue of the volumetric determination of the sulphur, however, is largely dependent upon the grade of the iron and the conditions under which the test is taken. A disturbing factor in establishing a satisfactory standard of comparative values in such analyses is the loss of uncertain and varying proportions of the sulphur, due to the formation of volatile and difficultly oxidizable organic sulphides, upon solution of white or chilled iron in hydrochloric acid. Comparative tests by a number of chemists of the Corporation having shown quite satisfactory agreement in sulphur-values as obtained by the evolution method in sand-cast and properly taken so-called chilled test-samples, it has been decided to adopt for all purposes the more convenient chilled test, which, it is believed, will tend to conserve the desired comparative accuracy. True values may be obtained in the crushed shot-sample by annealing the powder in a current of natural gas or under any convenient non-oxidizing conditions, but the exigencies of the routine of the works-laboratory, as demanded by the conditions necessarily involved in the manufacture of iron and steel, would in most cases prohibit this metallurgical refinement.

The methods of analysis herein described represent the best present-day practice and it is hoped that their adoption will go far towards reconciling those too often variant conditions, convenience of manipulation and accuracy of result.

The solutions as prescribed are indicated in their various proportions on a 1-liter basis. It will be found convenient, however, in most cases to employ in actual practice corresponding multiples of a greater volume.

III. Sampling.

1. *Character of the Sample.*

Two optional methods are prescribed for taking the test:

A. Plate or Pat Test.—With a suitable hand-ladle a portion is dipped from the stream of molten iron and, with the spoon of the ladle resting upon a clean dry iron plate, a pat of such size as may be desired is poured thereon.

B. Chilled-Mold Test.—With the spoon of the hand-ladle

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resting upon the dry clean mold (later to be described) a portion is poured sufficient to fill the mold.

The use of the water-shot sample is to be considered as in violation of the standard method. The sand-cast sample may be used, but the uniform adoption of the pat or the chilled-mold test is urged.

2. *Number of Samples to Be Taken.*

As tending to a reasonable degree of accuracy, it is recommended that a portion be secured, timed as nearly as may be practicable for the middle of each ladle of iron representing the cast. Equal portions from each of these samples are conveniently combined for the subsequent analysis.

3. *Preparation of the Sample.*

The tests are either drilled or crushed, as required. If crushed, only that portion passing an 80-mesh sieve is used for the subsequent analysis.

4. *Description of the Mold.*

Two convenient forms of mold are recommended, either of which may be used to advantage.

The mold, Fig. 1, furnishes a test which may be conveniently broken and crushed. The number of the furnace may be cast in the mold with Roman numerals or any other suitable form of designation. The two ridges shown in the test in this figure indicate that the test is taken from furnace No. 2.

The mold, Fig. 2, furnishes a test which may be either drilled or crushed. If the sample is to be crushed, the notched prolongation *T* is broken and crushed, and the main body of the test reserved for the furnace-man's fracture-test. The furnace-number may also be indicated in this form of mold in the same manner but not so conveniently, and only in the main body of the test.

The molds, made of cast-iron, are illustrated in detail, with complete dimensions, in Figs. 1 and 2.

IV. Methods of Analysis.

1. *Determination of Silicon.*

A. Drown's Method.—A single or double factor-weight, 0.4693 or 0.9386 g., of the sample is transferred to a glazed porcelain

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dish, from 15 to 25 cc. of the silicon mixture added, gentle heat applied to the uncovered dish, and the solution is evaporated to pronounced fumes of sulphuric anhydride. The dish is removed from the heat and allowed to cool, the residue is moistened with 10 cc. of the dilute hydrochloric acid, 50 cc. of hot water added, and the solution heated without boiling until the soluble salts dissolve. The residue of silica and graphite is collected on a 9-cm. ashless filter, washed alternately with hot dilute hydrochloric acid and water to the disappearance of the soluble iron, and finally with hot water until free from

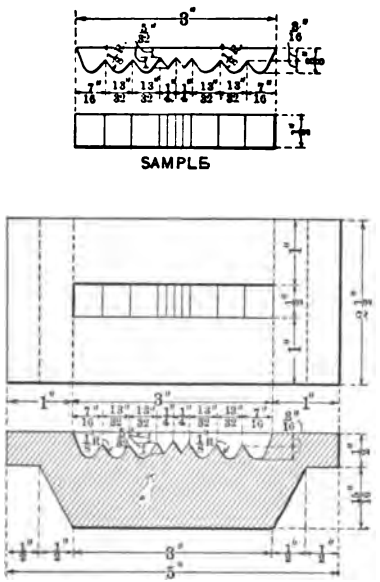

FIG. 1.—IRON-MOLD AND TEST.

chlorides. The residue is ignited uncovered in a weighed crucible at a moderate temperature until the carbon is completely oxidized, and finally for a few minutes at a higher temperature to the dehydration of the silica. The residual weight in milligrams divided by ten and corrected in conformity with the initial weight of the sample equals per cent. of silicon.

B. Ford's Hydrochloric Acid Method. — This well-known method is useful in point of rapidity, yielding satisfactory results in a shorter time than the more refined Drown's method. A single or double factor-weight of the sample is transferred

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to a porcelain or platinum dish with cover-glass, 20 or 30 cc. of concentrated hydrochloric acid added, and the solution boiled rapidly to complete dryness; the residue is not baked. Ordinarily, decomposition is complete by the time the evaporation

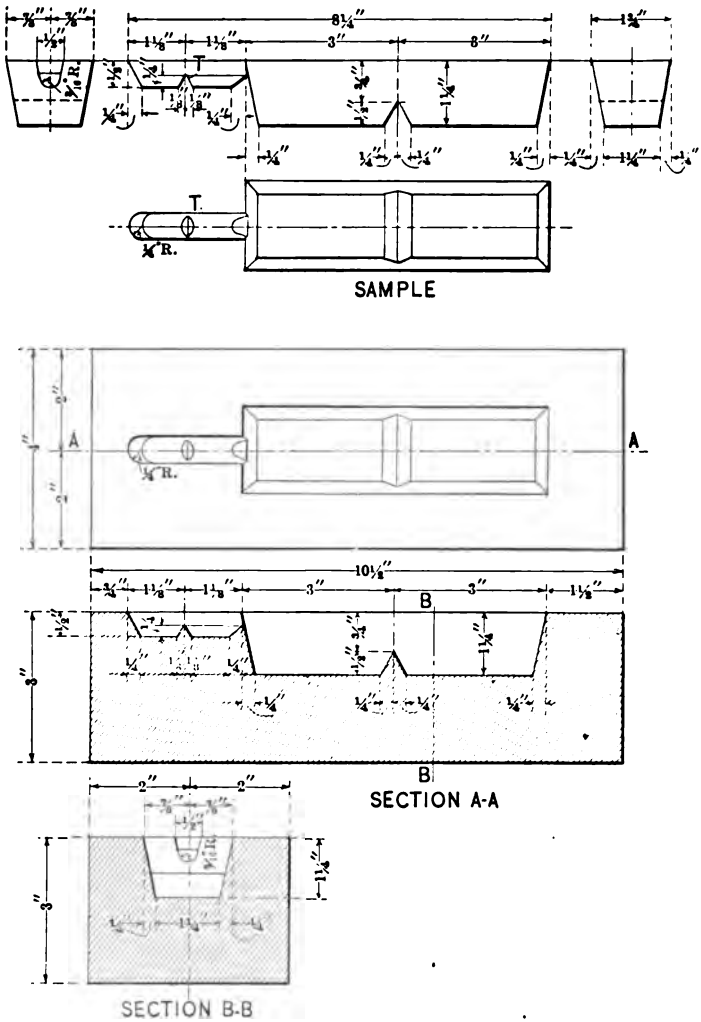

FIG. 2.—IRON-MOLD AND TEST.

is effected. To the dry residue in the platinum dish there is added at once, or in the porcelain dish when sufficiently cooled, 20 cc. of the dilute hydrochloric acid. The solution is heated gently for a few minutes with the addition of from 50 to 60 cc.

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of water, and the heating continued until the soluble salts dissolve. The residue is collected and finally weighed as in the previously described Drown's method. The use of suction in the filtrations and the ignition of the residue in an atmosphere of oxygen will be appreciated as factors affecting the rapidity of the analyses in both of the above methods.

SOLUTIONS REQUIRED.

Silicon Mixture.

Nitric acid	1.20 sp. gr.	750 cc.
Sulphuric acid	1.50 sp. gr.	250 cc.

Dilute Hydrochloric Acid.

Hydrochloric acid	1.20 sp. gr.	500 cc.
Water		500 cc.

2. *Determination of Sulphur.*

A. Volumetric Method. Evolution Iodine Titration.—It is apparent that the evolution method is of comparative value only and is in no sense to be regarded as yielding the exact measure of the sulphur-content of the iron, a fact of which due cognizance should be taken by all users of pig-iron. Equally apparent, however, is the undoubted usefulness of the method in connection with the routine requirements of the iron-laboratory and of the various metallurgical processes in conjunction with which it is used.

The apparatus as recommended for this determination consists of a flask of suitable form and size, preferably 16-oz. Florence ring neck, with a two-hole rubber stopper equipped with an ordinary funnel or thistle-tube and a short piece of glass tubing bent at right angles (exit-tube) in the respective apertures. The exit- or delivery-tube is connected with the absorption-vessel in such manner as to conduct the evolved gases to the bottom of the vessel. The absorption may be effected either in the beaker in which the titration is to be conducted, or in a test-tube, preferably 10 by 1 in., suitably supported.

A minimum of 2.5 g. or a maximum of 5 g. of the sample is transferred to the flask, the stopper is inserted and the flask conveniently connected with the vessel containing the absorbent; from 40 to 60 cc. of dilute hydrochloric acid is introduced through the funnel and heat applied, a brisk evolution of

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the gases being maintained until the powder is decomposed. The temperature is then raised until the steam formed has forced the last trace of the hydrogen sulphide gas into the absorbent. The heat is removed, the apparatus disconnected and the titration conducted.

Ten cc. of the starch solution is added to the titrating-beaker and the contents of the absorption-tube transferred thereto. Both the absorption- and delivery-tubes are rinsed with cold water and any adhering precipitate is dissolved by means of a fine jet of the dilute hydrochloric acid. Care is exercised that the acid solution be not unduly agitated before the introduction of the iodine solution. If the absorption is effected in the beaker, the delivery-tube is removed first, cleansed with the acid and water and, with the precaution suggested above, an excess of the dilute acid is added. The final volume of the solution should approximate 400 cc. The solution is stirred gently in order to neutralize the alkali of the absorbent, which, if left undisturbed, reacts with the iodine with formation of a substitution-product from which the iodine is but partly liberated upon acidifying the solution. The iodine solution is then added in successive portions, while the solution is stirred as the color fades, until the addition of a final drop yields the permanent blue color of the adopted end-point. The number of cubic centimeters of the iodine required multiplied by 0.01 is equivalent to the per cent. of sulphur.

The volume of the solution in the absorption-vessel is so adjusted as to insure a sufficient body for the absorption of all of the gas, and a sufficient excess of the stock absorbent is maintained in proportion to the total volume of the solution. The proportions as used in the 10- by 1-in. test-tube (15 cc. of any of the stock absorbents diluted to 60 cc.) may be taken as a basis therefor. If desired, the starch indicator solution may be added to any of the stock absorbent solutions in such proportions as to furnish the equivalent of 10 cc. of the indicator in the titration.

SOLUTIONS REQUIRED.

Starch Indicator.—To 1 liter of boiling water in a suitable flask, a cold emulsion of 6 g. of starch suspended in 100 cc. of cold water is cautiously added, and the boiling continued 5

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min. after the final addition. The solution is cooled to room-temperature, and, with the addition of 6 g. of zinc chloride in 50 cc. of cold water, mixed thoroughly and set aside for a day or two, with occasional shaking. After the final subsidence of the heavier residue, the starch solution is siphoned into the container and the settlings rejected. The zinc chloride is added as a preservative. Preliminary boiling of the water tends to lessen the frothing or formation of large bubbles on the addition of the starch.

Standard Iodine.—10 g. of potassium iodide crystals, followed by 4.5 g. of resublimed iodine, are transferred to a glass-stoppered graduated flask; 25 cc. of cold water is added and the flask is shaken until the iodine is completely dissolved. The solution is diluted to volume of 1 liter and set aside for not less than 24 hr. before standardizing. The known sulphur of a standard steel, as determined both by the evolution and oxidation methods, is used as the basis of standardization with final adjustment so that 1 cc. is equivalent to 0.01 per cent. of sulphur on the basis of the weight of sample used.

Dilute Hydrochloric Acid.—Concentrated acid, 500 cc.; water, 500 cc.

ABSORBENTS.

Ammoniacal Solution of Cadmium Chloride.

Cadmium chloride	5 g.
Water	375 cc.
Ammonia	625 cc.

Caustic Alkali.

Caustic potash or soda	5 g.
Water	1,000 cc.

Ammoniacal Solution of Zinc Sulphate.

Zinc sulphate	10 g.
Strong ammonia	625 cc.
Water	375 cc.

B. Gravimetric Method.—5 g. of the sample is transferred to a clean No. 4 beaker or 12-oz. Erlenmeyer flask, and, with the addition of 100 cc. of concentrated nitric acid, gentle heat applied until solution is complete. It is sometimes necessary to check a too violent action of the acid, which might result in loss of unoxidized sulphur, by placing the beaker or flask in cold water. The solution is boiled briskly to low bulk, and

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after cooling somewhat, 30 cc. of strong hydrochloric acid added, the solution is again concentrated, and following the precautionary addition of 1 g. of pure sodium carbonate, transferred to a clean porcelain dish and evaporated to complete dryness and the residue baked a short time at a moderate temperature.

The dish is allowed to cool, 35 cc. of concentrated hydrochloric acid added and heat applied until the soluble salts dissolve; the solution is again evaporated to complete dryness and the residue baked as before. This procedure insures the elimination of the nitric acid. The dish is cooled, 35 cc. of concentrated hydrochloric acid is added and heat is applied until the soluble salts are again dissolved. The cover is removed, and with the dish placed upon a suitable asbestos ring in order to prevent the formation of a crust of the metallic salts upon the sides, the solution is evaporated to the first appearance of the separation of the chlorides, which are then dissolved by the addition of a few drops of concentrated hydrochloric acid; three drops should suffice. Cold water is added to volume of about 75 cc., the residue is collected upon an 11-cm. filter and the filtrate received in a No. 2 beaker. The residue and filter are washed with the least possible amount of the hot dilute hydrochloric acid, alternated with cold water, to the disappearance of the iron stains and finally with hot water to volume of about 200 cc. To this cold solution 10 cc. of a 10 per cent. solution of barium chloride is added and the solution is allowed to stand, with occasional stirring, not less than 12 hours.

When the precipitate of barium sulphate has subsided, it is collected upon a 9-cm. close ashless filter, washed with dilute hydrochloric acid (5 per cent.) to the disappearance of the iron, and finally with hot water until free from chlorides. The filter and residue are ignited uncovered in a weighed crucible at a moderate temperature to the disappearance of the carbon of the paper, and finally at a somewhat increased temperature for 10 min. The residual weight multiplied by 13.74 and divided by 5 equals the per cent. of sulphur. The filtrate is set aside until satisfied that no further precipitation ensues.

The carbonaceous siliceous residue of the original solution usually retains some sulphur, and is ignited in platinum under

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cover of pure sodium carbonate, and decomposed by fusion with further addition of the carbonate and 0.2 g. of niter; the subsequent hydrochloric acid solution is freed from its silica and added to the original filtrate, or examined separately for sulphur.

It is apparent that absolute purity of reagents and cleanliness of utensils, and the final weighing of the barium sulphate effected under the most exact conditions, are determining factors in the successful issue of analyses by this method.

3. Determination of Phosphorus.

A. Volumetric Method. Acid-Alkali Titration.—From 1 to 2 g. of the sample is transferred to a flask, preferably 12-oz. Erlenmeyer, 40 cc. of nitric acid (1.13 sp. gr.) added, and heat applied until the sample is decomposed. The sides of the flask are washed down, 1 g. of ammonium persulphate is added, and the solution boiled until the combined carbon is completely oxidized. The residue is caught on an 11-cm. filter and washed with dilute nitric acid (2 per cent.) and hot water to the disappearance of the soluble iron. The filtrate is heated to boiling in a suitable flask, a slight excess of a solution of potassium permanganate added, and the boiling is continued until the excess of the permanganate is decomposed. The resultant precipitate of manganese dioxide is reduced to its soluble nitrate by means of the chosen reducing agent and the boiling continued until the excess thereof is decomposed.

With the solution at a temperature of 80° C., from 40 to 60 cc. of ammonium molybdate solution is introduced and the flask is shaken continuously for 5 min. The precipitate quickly subsides and is at once collected upon a 9-cm. filter, washed with dilute nitric acid (2 per cent.) until free from iron, and then with solution of potassium nitrate (1 per cent.) to the removal of the last trace of free acid.

The titration may be conducted in a beaker or in the flask in which the precipitation was effected. Care is exercised in the latter case to wash the flask thoroughly with the neutral wash, as any acid present of course reacts with the standard alkali. The filter and precipitate are transferred to the beaker or flask and the standard alkali is added in excess sufficient to render easy the subsequent maceration of the filter, which is accom-

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plished by breaking up with a clean glass rod, or, if in a flask, by shaking the flask closed by a clean rubber stopper. When the precipitate has dissolved and the paper is thoroughly disintegrated, the solution is diluted and three drops of the indicator is added. The standard acid is then added to the disappearance of the pink tint, the sides of the titrating-vessel washed down with cold water, and, with a final volume of about 150 cc., the standard alkali is added to the adopted pink end-point. The number of cubic centimeters of the alkali used in excess of the acid, multiplied by 0.01 or 0.02, is the equivalent of phosphorus in hundredths per cent.

SOLUTIONS REQUIRED.

Standard Caustic Soda.—6.6 g. of pure sodium hydroxide is dissolved in 100 cc. of boiled water. The solution is freed from any carbonic acid introduced by the caustic alkali, by the addition of about 1 cc. of a saturated solution of barium hydroxide, with subsequent filtration, and the solution finally diluted to volume of 1 liter. If desired, a stock solution may be carried, a convenient aliquot portion of which is diluted for the working-solution.

Standard Nitric Acid.—10 cc. of concentrated acid diluted to 1 liter. The acid and alkali solutions are standardized against the known phosphorus of a standard steel, with final correction so that they are actually agreeing and 1 cc. is equivalent to 0.01 per cent. of phosphorus on the basis of 2 g. of the sample.

Ammonium Molybdate.—To 65 g. of molybdic acid (85 per cent.), suspended in 142 cc. of cold water in a suitable flask, 143 cc. of strong ammonia is added, and the flask is rotated until the molybdic acid is completely dissolved. The solution is cooled and added slowly to 715 cc. of nitric acid of 1.20 sp. gr. while the acid solution is vigorously stirred.

Molybdic acid is at times encountered which fails to yield a permanently clear solution of the ammonium molybdate; the addition of one or more drops of a 10 per cent. solution of ammonium phosphate assists in clarifying the solution. The solution is allowed to stand over night, when it is filtered into the container, preferably through purified asbestos and with the aid of suction.

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Phenolphthalein.—1 g. dissolved in 1 liter of ethyl alcohol.

Nitric Acid (1.13 sp. gr.).—Nitric acid, concentrated, 240 cc.; water, 760 cc.

Potassium Permanganate Oxidant.—25 g. to 1 liter.

REDUCING AGENTS.

Ammonium Bisulphite.—5 per cent. solution.

Ferrous Sulphate.—Ferrous sulphate, 50 g.; water, 1,000 cc.

Potassium Nitrite.—Potassium nitrite, 50 g.; water, 1,000 cc.

Sugar.—Saturated solution.

B. Volumetric Method.—Potassium Permanganate Titration.—The yellow precipitate as above obtained is washed with ammonium sulphate solution, or, if desired, dilute sulphuric acid (2 per cent.) until free from iron and molybdenum salts. Dilute ammonia (1 part ammonia and 3 parts water) is poured over the precipitate and the filter is washed with hot water, and the solution caught in the flask in which the precipitation was effected. The volume of the solution should not exceed 25 cc. 8 g. of granulated zinc, 20-mesh, followed by 75 cc. of dilute sulphuric acid, is added, and the solution is gently boiled until reduction is complete. The undissolved zinc is then collected upon a glass-wool filter and both flask and filter are washed with cold water. To the cold filtrate, standard potassium permanganate solution is added until a slight permanent pink tint is obtained. The measure of the permanganate solution required multiplied by its value in terms of phosphorus and divided by the initial weight of the sample equals the per cent. of phosphorus.

SOLUTIONS REQUIRED.

Acid Ammonium Sulphate.

Strong ammonia	15 cc.
Sulphuric acid (1 part acid to 1 part water)	50 cc.
Diluted to 1 liter.	

Standard Potassium Permanganate.—2 g. of potassium permanganate dissolved in 1 liter of water; 1 cc. equals approximately 0.0056 per cent. of phosphorus on a 1-g. basis. The known phosphorus of a standard steel is used as the basis of standardization.

C. Gravimetric Method, Weighing as Ammonium Phospho-

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molybdate.—From 1 to 5 g. of the sample is transferred to a No. 5 glazed porcelain dish with cover-glass, from 25 to 60 cc. of nitric acid (1.13 sp. gr.) is cautiously added, gentle heat is applied until the sample is completely decomposed, and the solution is then rapidly boiled to dryness. The temperature is raised gradually with final heating of the uncovered dish over the full flame of an Argand or Bunsen burner to the complete expulsion of the acid. After cooling sufficiently to avoid danger of breaking the dish, a minimum of 30 cc. of concentrated hydrochloric acid is added, heat is applied until the soluble salts are dissolved, and the solution rapidly boiled to the first appearance of the separation of the chlorides. This evaporation is conducted with the same regard to the prevention of the formation of a crust upon the sides of the dish as in the previously described method for the determination of sulphur. The dish is removed from the heat, 10 cc. of strong nitric acid added, and heat again applied for about 1 min. Cold water is added to volume of about 60 cc., the solution stirred with a clean glass rod and filtered into a suitable flask; the residue, collected upon an 11-cm. filter, is washed with 2 per cent. nitric acid and hot water alternately until free from iron, which insures also the complete removal of the soluble phosphate.

When the iron contains titanium in amount sufficient to interfere with the precipitation of the phosphorus and an exact determination is required, the residue of silica and graphitic carbon is ignited in a platinum crucible, the silica volatilized with hydrofluoric acid in the presence of a few drops of sulphuric acid, and the resultant residue ignited and fused with sodium carbonate. The melt is boiled with hot water until disintegrated, the insoluble titanate is collected upon a filter and the soluble phosphate added to the main solution.

To the clear filtrate, with the volume of about 150 cc., 25 cc. of strong ammonia is added at once, and the flask is shaken until the resultant precipitate has coagulated. The addition now of 25 cc. of strong nitric acid renders the solution clear and furnishes an excess of about 10 cc. of strong nitric acid, which, it is found, yields a pure crystalline precipitate of the phosphomolybdate with no tendency to creep. The relatively large amounts of ammonia and nitric acid added are prescribed in the case of 5-g. samples, and are essential in that they insure

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a sufficient proportion of ammonium nitrate in the solution. It is evident that these additions may be decreased with smaller initial weights of the sample, but it will be found advantageous to maintain approximately the excess of 10 cc. of the nitric acid.

With the solution at a temperature of 80° C., from 50 to 75 cc. of the molybdate solution is introduced and the solution agitated for about 5 min. When the yellow precipitate has finally subsided, it is collected upon a previously dried and weighed 9-cm. filter, the filter and contents are washed thoroughly with 2 per cent. nitric acid, dried for 1 hr. in an air-bath at a temperature of about 115° C. and weighed between watch-glasses with ground edges and inclosed in a suitable clip. It is of course essential that the light weight of the previously-dried paper shall have been obtained under the same conditions. The increase in weight multiplied by 1.63 and divided by the initial weight taken is equivalent to the per cent. of phosphorus.

D. Gravimetric Method, Weighing as Magnesium Pyrophosphate.—The yellow precipitate as above obtained is dissolved on the filter with hot ammonium citrate solution and the filter thoroughly washed with hot water, and the solution retained in a No. 0 beaker. To this cold solution, from 5 to 10 cc. of the magnesia mixture is added, the solution is well stirred and allowed to stand in the cold not less than 3 hr. The precipitate of magnesium phosphate is collected upon a 9-cm. close ashless filter, washed with dilute ammonia, ignited at a low temperature to the disappearance of the carbon of the filter and finally for 10 min. at the full temperature of the blast. The weight of the magnesium pyrophosphate multiplied by 27.87 and divided by the initial weight taken is equivalent to per cent. of phosphorus.

SOLUTIONS REQUIRED.

Ammonium Citrate.

Citric acid	50 g.
Strong ammonia	350 cc.
Water	650 cc.

Magnesia Mixture.

Magnesium sulphate	84 g.
Ammonium chloride	250 g.
Water	665 cc.
Ammonia	335 cc.

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Dilute Ammonia.

Strong ammonia	200 cc.
Water	800 cc.

4. *Determination of Manganese.*

A. Walters's Color-Comparison Method.—From 50 to 200 mg. of the sample is transferred to a suitable tube or flask, preferably an 8- by 1-in. test-tube or a 2-oz. Erlenmeyer flask, 10 cc. of nitric acid (1.20 sp. gr.) added and heat applied either in a water-bath or on a hot-plate until the sample is decomposed. About 0.5 gr. of moist ammonium persulphate, or its equivalent solution, is added and the heating continued until the combined carbon is completely oxidized. If silica or graphitic carbon separate in amount sufficient to render it necessary, the solution is filtered into a similar vessel and the residue retained on a small filter. The filter and residue are washed with the minimum of water, or the 15 cc. of silver nitrate solution required for the subsequent oxidation. The silver nitrate is introduced either through the filter or into the unfiltered solution, 1 g. of moist persulphate is added and the heating continued until the characteristic pink tint of the permanganic acid is well defined. The solution is cooled, transferred to the comparison-tube and the color matched against a suitable standard which has been treated in exactly the same manner, and with the same relative initial weight. The relative volume of the solution as required to match the color of the standard is equivalent to hundredths per cent. of manganese.

B. Volumetric Method.—The solution of permanganic acid is transferred to a beaker or flask, and following the addition of 10 cc. of sodium chloride solution, the standard sodium arsenite is added to the definite end-point of the disappearance of the pink tint. The volume of the solution should not exceed 75 cc. If the silver is not precipitated, the reaction of the persulphate with the silver nitrate renders the end-point somewhat fleeting and recurring. If conducted without delay, however, the titration may be accomplished with perfect satisfaction. The end-point, in this case, while not so permanent, is perhaps somewhat sharper than in the presence of the chloride.

When desired, a greater initial weight of the sample may be taken, the solution thereof diluted to a definite volume, the

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graphite and silica allowed to subside, and aliquot portions withdrawn with a pipette.

SOLUTIONS REQUIRED.

Sodium Chloride.—2.5 g. of sodium chloride to 1 liter of water.

Silver Nitrate.—About 66.5 g. of silver nitrate is dissolved in 1 liter of water; 20 cc. of this diluted to 1 liter affords a solution of which 15 cc. is equivalent to approximately 0.02 g. of silver nitrate.

Sodium Arsenite.—A stock solution is prepared by dissolving 10 g. of c.p. arsenious acid in a boiling aqueous solution of sodium carbonate (30 g. dissolved in 250 cc. of water), with final dilution of the cold solution to volume of 1 liter. 62.5 cc. of this solution is diluted to volume of 1 liter for the working-solution, of which each cubic centimeter is equivalent to approximately 0.1 per cent. of manganese, with an initial weight of 0.2 g. of the sample. The known manganese of a standard steel or iron is used as the basis of standardization.

C. Ford's Gravimetric Method.—5 g. of the sample is transferred to a No. 5 glazed porcelain dish with cover-glass, 75 cc. of nitric acid (1.20 sp. gr.) cautiously added and gentle heat applied until the sample is decomposed, and the solution then rapidly boiled to complete dryness. The residue is baked a short time at a moderate temperature. The dish is cooled, 35 cc. of strong hydrochloric acid is added and heat is applied until the soluble salts dissolve. Cold water is added to approximately 60 cc., the solution is filtered into a No. 5 beaker and the residue caught on an 11-cm. filter and washed with the least amount necessary of hot dilute hydrochloric acid and cold water alternately, to the disappearance of the soluble metallic salts. The filtrate is boiled to low bulk, cooled, 50 cc. of strong nitric acid is added and the solution is again boiled down to low bulk. The evaporation is repeated with successive portions of nitric acid to the complete expulsion of the hydrochloric acid and with conversion of the chlorides to their corresponding nitrates; the evaporation at this point is carried to the first indication of the separation of the nitrates as observed in the formation of a small nucleus of scum floating on the surface of the solution. After the solution has cooled somewhat, 75 cc. of strong nitric acid is added and the solution is again brought to the boiling-point. To the boiling

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solution, crystals of potassium chlorate (5 g. will be found sufficient) are cautiously added in small portions from a glass spoon, and the solution is boiled for 5 minutes.

The solution is allowed to cool before filtering. Filtration is effected through the medium of an asbestos plug held in a carbon filtering-tube. The asbestos is prepared for use by heating with aqua regia to remove soluble lime salts and the plug is washed with strong nitric acid before using. The precipitated manganese dioxide on the filter is then washed with successive additions of strong nitric acid (free from nitrous fumes) to the practical elimination of the iron; the acid is added first to the beaker and transferred from thence to the filter. After the final washing, suction is continued until the plug is practically dry, when the rubber stopper carrying the filter is transferred to a clean flask. The filtering-tube is filled with 1-per cent. solution of ammonium bisulphite and allowed to stand about 1 min., suction is then applied, successive portions of the bisulphite solution being added until the last trace of manganese dioxide is dissolved; the soluble manganese salt is finally washed from the asbestos filter with hot water. The solution is transferred to a No. 4 beaker or, if preferred, retained in the flask, 10 cc. of strong nitric acid added and the excess of the bisulphite decomposed by boiling. The manganese dioxide could be dissolved perhaps more conveniently by means of hot hydrochloric acid, but the use of the bisulphite solution obviates the possible danger of contamination by traces of soluble lime or magnesia, which in some grades of asbestos seem to be in evidence notwithstanding numerous successive treatments with the hot acid.

The solution is cooled somewhat and ammonia is added to the point of a faint permanent precipitate, followed by the addition of 15 cc. of a 20-per cent. solution of ammonium acetate. The solution is boiled 2 min., filtered into a No. 5 beaker and the precipitate of the basic acetate of iron collected on an 11-cm. filter. The beaker or flask and the filter are washed twice with hot water, the precipitate is dissolved with hot dilute nitric acid and the solution returned to the vessel in which the precipitation was effected. The filter is washed with hot water until free from iron and the basic acetate precipitation and subsequent filtration repeated.

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To the combined filtrates 5 cc. of glacial acetic acid is added and the solution is again brought to the boiling-point; the cover-glass is removed, 10 cc. of a 10-per cent. solution of ammonium or sodium phosphate added, and with the solution still at the boiling-point and with constant stirring, strong ammonia is added to the first appearance of an incipient opalescence. The stirring is continued until the opalescence has developed into the characteristic crystalline precipitate, at which point the addition of the ammonia is continued, drop by drop and with constant stirring, until the total of 25 cc. has been added.

The precipitate is filtered at once upon an 11-cm. ashless filter and washed with dilute ammonia water (2 per cent.) until free from soluble salts. The filter and residue are ignited in a weighed porcelain or alundum crucible at a low temperature until the paper is burned off, and finally at a somewhat higher but still moderate temperature for 10 minutes.

$$\frac{\text{Residual weight} \times 38.69}{5} = \text{per cent. of manganese.}$$

5. *Determination of Total Carbon.*

A. Direct Combustion Method.—The direct combustion of the carbon of iron or steel and their various alloys is fast coming to supersede the more tedious wet method. Certain limitations of this most excellent method, however, are to be noted. The inherent difficulty of successfully burning particles of steel of too great mass is apparent. Certain samples of both iron and steel or their alloys have been encountered of such a character as to necessitate the intimate admixture therewith of a finely-divided metallic oxide, or, in the case of certain irons, fine drillings of low-carbon steel, to assist in the oxidation. The powder of the crushed test of white iron requires a somewhat longer time for complete combustion than the drillings from the test of gray iron, but with proper care may be burned with perfect satisfaction. An essential requirement in the conditions of the method is the close control of the temperature, which should approximate a constant of 1,000° C., and should not fall below 960° C. A pyrometer-couple will be found convenient in controlling the temperature.

The combustion is conducted preferably in a fused-quartz

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tube in either an electric or a suitable gas-fired furnace. A platinum tube may be used, but is subject to damage from possible splashing of the oxidized metal, and is, moreover, much more expensive than the quartz. The use of litharge or other volatile and easily reducible oxides is, of course, prohibited in the platinum tube. The quartz tube is 24 in. long by 0.75 or $\frac{1}{8}$ in. in inside diameter, free from seams and selected of uniform diameter. Platinized asbestos, which may be secured at either end by short plugs of platinum gauze, is inserted for a space of 5 or 6 in. in the tube immediately following the position to be occupied by the boat. The rubber stopper at the end of the tube may be protected by the insertion of a removable plug of asbestos conveniently inclosed in a thin roll of platinum gauze. Platinum, nickel, alundum, porcelain, or clay boats may be used. The dimensions, from 3 to 3.5 in. long, by 0.25 in. deep, 0.5 in. wide at the bottom, flaring to $\frac{5}{8}$ in. at the top, all inside measurements, will be found convenient. Suitable furnaces for either direct or alternating current, with transformers, rheostats, etc., to suit the individual requirements, may be obtained of the trade.

The oxygen from the high-pressure cylinder under control of the reducing-valve, is purified by passing through caustic potash solution of 1.27 sp. gr. contained in washing-bottles, preferably two arranged in tandem. A glass tube of convenient form and size, filled with granulated zinc, may be connected to the exit end of the combustion-tube to retain possible fumes of sulphuric anhydride formed in the combustion of high-sulphur irons. When the potash absorbent is used, a calcium chloride drying-tube preceding the bulb or bottle is necessary.

Factor-weights may conveniently be used in the direct combustion as follows: For the potash-absorption and weighing of the bulb, 0.2727 g. or 0.5454 g., where each 0.1 mg. gain in weight of the bulb is equivalent to 0.01 per cent. or 0.005 per cent. of carbon respectively. For the barium hydrate absorption, with ignition of the barium carbonate, 0.304 g. or 0.608 g., where each 0.1 mg. of the weight of barium carbonate is equivalent to 0.002 per cent. and 0.001 per cent. of carbon respectively. For the titration methods it is convenient to employ

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for the initial weight even gram-multiples, preferably 250 mg. or 500 mg.

The 0.25 or 0.5 g. or the suitable factor-weight of the sample is transferred to the boat. The boat is partly filled with ignited alundum depressed in the center in the form of a V about 1 in. long, and in which the sample is spread in a compact mass. This intimate contact of the particles of iron conserves the heat for the continuous combustion of the carbon. The weighed potash-container or the Meyer bulb-tube, to which has been added from 80 to 100 cc. of the barium hydroxide solution or 50 cc. of the standard potash solution, is connected with the tube, and the boat is carefully pushed to its position in the center of the heat-zone. The stopper of the tube is replaced, and, with all connections tight, the current of oxygen is started at a rate approximating a continuous bubbling, as observed in the purifying-solutions. It is essential that the tube should have acquired the maximum temperature before the introduction of the boat, a condition which will naturally obtain in the case of a continuous routine of combustion.

The first indication of the oxidation is observed in the sudden decrease of the bubbling of the oxygen through the absorption-vessel. As the rate decreases, the flow of oxygen is somewhat accelerated until the conclusion of the oxidation process, as indicated by the recurrence of the rapid flow through the exit; the flow of oxygen is then reduced to the normal rate, which is maintained for 10 min. to insure the complete oxidation of all the carbon and the final absorption of the resultant carbon dioxide. The absorption-vessel is disconnected, the current of oxygen discontinued, and the analysis concluded by either weighing the potash bulb; titrating the standard solution of caustic potash; filtering, igniting, and weighing the precipitated barium carbonate; or titrating the excess of the barium hydrate solution with a weak solution of hydrochloric acid and without preliminary filtration.

By Increase of the Weight of the Potash-Bulb or Bottle.—The perfectly clean and dry bulb or bottle is at once weighed. The increase in weight as noted in connection with the initial weight of the sample is corrected to the per cent. of carbon.

The potash-container is prepared for use by carefully introducing the required volume of the solution, connecting with

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the apparatus and passing a current of oxygen through the hot combustion-tube to the complete displacement of the air in the container. The vessel is disconnected and at once weighed. The weight so obtained furnishes the light weight for the succeeding combustion.

Titration of the Caustic Alkali.—The absorption in this case is effected in a Meyer bulb-tube containing 50 cc. of the potash solution, accurately measured. The contents of the tube are transferred to a flask, the tube is cleansed with freshly boiled water, three drops of the phenolphthalein solution are introduced and dilute hydrochloric acid (1 part acid to 1 part water) carefully added until the color begins to fade. The more dilute standard acid is then added until the addition of one drop just discharges the pink tint. The excess alkali which has not entered into combination with the carbon dioxide has now been neutralized and the solution is ready for the actual titration. One drop of the methyl orange or methyl red solution is added and the standard acid added in successive portions until the addition of one drop causes the characteristic change from the yellow to the rose tint in the solution. The measure of the acid required for this titration minus the equivalent required in the blank titration and multiplied by the carbon-value of the solution equals per cent. of carbon.

If preferred, the titration may be conducted with the phenolphthalein indicator, the measure of the standard acid required to discharge the pink tint being deducted from the blank equivalent and calculated to per cent. of carbon as above. This method, however, would not appear to offer any advantage over the other and obviously requires more of the standard acid.

The blank equivalent is established by passing the oxygen through the hot combustion-tube and into the measured volume of the absorbent solution with final titration as outlined above. The carbon-value of the acid solution is established by titration of the measured volume of the potash solution following the combustion of a standard steel. Additional confirmation may be had by titrating against a standard solution of sodium carbonate. The use of sodium oxalate in this connection is recommended.

SOLUTIONS REQUIRED.

Standard Acid, One-Sixth Normal.—15 cc. of hydrochloric acid (1.20 sp. gr.) diluted to 1 liter.

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Caustic Potash, One-Sixth Normal.—9.5 g. diluted to 1 liter. This solution need not of necessity be absolutely one-sixth normal.

Phenolphthalein.—1 g. dissolved in 1 liter of ethyl alcohol.

Methyl Orange.—1 g. dissolved in 1 liter of water.

Methyl Red.—1 g. dissolved in 1 liter of ethyl alcohol.

Filtration and Ignition of the Barium Carbonate.—The bulb-tube is detached from the train, the index finger of the left hand being simultaneously placed over the exit end in order to prevent the solution from flowing back in the tube. The solution is filtered through an 11-cm. ashless filter, the filter and precipitate are washed with freshly-boiled water (six good washings will be found sufficient), ignited in a weighed platinum crucible at a moderate temperature to the disappearance of the carbon of the filter and finally at a good full heat for 5 min. The weight of the barium carbonate multiplied by 6.08 and divided by the weight taken, or with correction in agreement with the initial factor-weight, is equivalent to the per cent. of carbon. The solution and precipitate are transferred to the filter either by way of the bulb-entrance with control of the flow by the index finger upon the open exit end of the tube or by control with the thumb placed over the bulb entrance with the solution poured through the exit end. The precipitate may be readily removed by shaking the tube after successive additions of the water. The washing is conducted as thoroughly and expeditiously as possible and care is exercised to avoid the passage of air unduly through the filter.

Titration of the Excess Barium Hydroxide. The bulb-tube is detached as before, the solution and precipitate are transferred to a 12-oz. Erlenmeyer flask, and the tube thoroughly washed with freshly-boiled water. Three drops of the indicator solution is introduced and the standard hydrochloric acid is added in successive small portions, while the flask is shaken, to the disappearance of the pink tint. The agitation of the solution insures the recovery of any carbon dioxide liberated upon the addition of the acid. The equivalent of the acid required in a preliminary blank titration minus the measure required in the actual titration, multiplied by the factor of the solution as determined by frequent combustions of a standard steel and in agreement with the initial weight of the sample, is equivalent to the per cent. of carbon.

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SOLUTIONS REQUIRED.

Standard Hydrochloric Acid.—8 cc. of strong hydrochloric acid diluted to 1 liter. Freshly-boiled water should be used in making up the solution. 1 cc. of this solution is equivalent to approximately 0.05 per cent. of carbon on a 1-g. basis.

Barium Hydroxide.—20 g. of barium hydroxide crystals are dissolved in 1 liter of hot freshly-boiled water and the flask covered and set aside to cool. When the solution has reached the room-temperature, it is filtered as rapidly as possible through a 25-cm. filter into the container. The solution is preferably withdrawn from the container by means of an overflow pipette in conjunction with a suitable guard-tube filled with soda lime or caustic potash solution.

Phenolphthalein.—1 g. dissolved in 1 liter of ethyl alcohol.

B. Solution and Combustion Method.—The conditions involved in the operation of this method admit of the employment of a greater initial weight of sample than in the previously described direct combustion method.

One g., or if the barium hydroxide absorbent is to be used, a ten factor-weight (0.608 g.) of the sample is transferred to a No. 2 beaker or 4-oz. Erlenmeyer flask; 100 cc. of the solution of copper and ammonium chloride or the potassium salt is added and the covered beaker is placed on a suitable stirring-machine or stirred by hand, at a temperature of 60° to 70° C., until in solution; or, if a flask is used, it is placed on a shaking-machine or closed with a clean rubber stopper and shaken by hand until solution is complete.

The carbonaceous residue is collected upon a filter of purified ignited asbestos made up in a perforated platinum boat, Gooch crucible, or other suitable form of holder. The residue is washed by decantation with dilute hydrochloric acid to the disappearance of color in the washings, then transferred to the filter and washed with cold water until the acid is completely removed. Preliminary drying of the residue is not essential in routine work, but where the refinement of accuracy is desired it is recommended that the residue be dried in a hot-air or steam bath at a temperature of 100° to 105° C. The residue is ignited in a platinum or silica tube, combustion-crucible, or other form of apparatus, and in an atmosphere of pure oxygen or air free from carbon dioxide. The apparatus and the sub-

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sequent treatment of this carbonaceous residue may conveniently conform to the description of the method by direct combustion, and the final measure of the carbon be established by any of the several optional methods therein described.

SOLUTIONS REQUIRED.

Copper and Ammonium Chloride.—300 g. of the pure salt is dissolved in 1 liter of boiling water. After cooling, 60 cc. of strong hydrochloric acid is added and the solution is filtered through a purified asbestos plug into the container.

Copper and Potassium Chloride.—1 lb. dissolved in 1 liter of water, as above, with subsequent similar treatment.

Caustic Potash (1.27 sp. g.).—300 g. dissolved in 1 liter of water.

6. *Determination of Graphitic Carbon.*

One g. or the ten factor-weight (0.608 g.) is transferred to a No. 2 beaker or a 4-oz. Erlenmeyer flask, 40 cc. of nitric acid (1.13 sp. gr.) added and gentle heat applied until solution is complete. The solution is then boiled for a few minutes, the residue is collected upon a suitable asbestos or paper filter, washed alternately with hot dilute hydrochloric acid and water until free from iron, and finally with hot water until free from acid. If the filtration has been conducted through an asbestos filter, the residue may be washed with hot caustic potash (1.10 sp. gr.) to remove separated silica. If preferred, a few drops of hydrofluoric acid may be added to the acid solution to dissolve the silica, which might otherwise interfere with the filtration.

The analysis may be concluded in either of the following optional ways:

1. The graphitic residue is collected upon a suitable filter, washed with dilute hydrochloric acid and water, and ignited in the combustion-apparatus, and the measure of the graphitic content established as in the case of total carbon. It will be found convenient in this case to use the ten factor-weight of the sample.

2. The filter and graphitic residue are placed in a crucible and dried to constant weight at a temperature of 110° C., then ignited until the graphitic carbon and the filter-paper are

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completely oxidized, and again weighed. The loss in weight minus the weight of the filter-paper when used and multiplied by 100 is equivalent to per cent. of graphitic carbon on a 1-g. sample.

7. *Determination of Combined Carbon.*

A. Indirect Method.—The per cent. of graphitic carbon as found above subtracted from that of the total yields the measure of the combined carbon. If desired, a direct estimation of the combined carbon may be conducted by the color-comparison method.

B. Color-Comparison Method.—The relatively exact chemical characteristics and preliminary heat-treatment of the test-piece and standard are universally recognized as fundamental bases of accuracy in the determination of the combined carbon of iron or steel by the color-comparison method. The universal and continued use of this most convenient method in the analysis of steel-products attests its efficiency within the close limits of control possible of attainment in the preparation of the steel standards. The application of the method to the determination of the combined carbon of the various grades of iron is much more limited, owing to the larger and varying content of manganese and sulphur, the principal disturbing chemical factors affecting the color of the solution to be compared. It is apparent, however, that with a judicious selection of standards and careful control of all factors involved, the color-comparison method may be applied to the determination of the combined-carbon content of iron with a quite reasonable degree of accuracy.

The method of procedure as recommended is as follows: 1 g. of the sample is transferred to a dry clean beaker, flask, or tube, as desired. In the presence of graphitic carbon, the use of a magnetized spatula or wire in weighing off the sample is, of course, prohibited. 30 cc. of nitric acid (1.20 sp. gr.) is added and heat is applied until the sample is decomposed. Graphitic carbon and silica are removed by filtration, and the filtrate retained in a 100-cc. graduated flask or cylinder. The filter is washed with cold water to the disappearance of color; the solution is diluted to volume, thoroughly mixed, and 10 cc. withdrawn with a pipette and transferred to the comparison-tube. The color is then compared with the color of the solution of a

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standard iron in a similar tube and which has been treated in exactly the same manner and with the same relative initial weight. The simple calculation involved in the relative volumes of the two solutions yields the per cent. of carbon in the sample.

A convenient aliquot portion of the above solution may be employed for the determination of manganese by the persulphate color method.

8. *Determination of Titanium.*

Five g. of the sample is transferred to a No. 5 glazed porcelain dish or to a No. 4 beaker, with cover-glass; 50 cc. of strong hydrochloric acid is added and the sample is digested until completely decomposed. Practically all of the titanium is said to remain insoluble with the silica and graphitic carbon. The insoluble residue is collected upon an 11-cm. filter, washed a few times with hot water, ignited in a platinum crucible and the silica volatilized with hydrofluoric acid in the presence of a few drops of sulphuric acid. The residue is then carefully ignited to the complete expulsion of the hydrofluoric acid, which, if retained, prohibits the subsequent colorimetric determination of the titanium.

Should it be desired to determine the usually quite negligible amount of titanium in the filtrate from the insoluble residue, it is recovered as follows: The filtrate in a No. 5 beaker is diluted to 250 cc. and strong ammonia added to the first appearance of a precipitate which slowly dissolves upon stirring. Dilute ammonia (2 per cent.) is added to a faint permanent precipitate, which is then dissolved by the addition of 15 cc. of a 10-per cent. solution of hydrochloric acid. The precipitate thus formed should dissolve rather slowly following vigorous stirring of the solution. 100 cc. or more of a 20-per cent. solution of sodium hyposulphite is added and the solution in the beaker is stirred until the iron is completely reduced and the free sulphur begins to separate. The solution is then boiled for 10 min. and the precipitated titanic acid allowed to subside and collected upon an 11-cm. filter. The filter and precipitate are washed free from soluble salts with dilute acetic acid (2 per cent.) and ignited in the crucible containing the ignited residue from the previously-conducted volatilization.

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The combined residues are fused with 4 g. of sodium carbonate, the melt is disintegrated by boiling with hot water, and the insoluble sodium titanate is collected upon a 9-cm. filter and washed with hot water containing a little sodium carbonate. Hot dilute sulphuric acid is added to the crucible, which is then heated until the titanate is completely dissolved. The filter is spread upon the bottom of a No. 2 beaker, the contents of the crucible transferred thereto, and the crucible washed thoroughly with hot water. The beaker is then heated until the titanium salt dissolves, when the paper is removed and its burden of titanium sulphate returned to the beaker by washing with hot water. The solution is then transferred to the color-comparison tube, or if necessary to a graduated flask, and aliquot portions taken. Shreds of filter-paper which may be observed in the solution are removed by filtration.

In the color comparison, 5 cc. of c.p. hydrogen peroxide is added to the solution in the comparison-tube and the solution is diluted to a volume yielding a color convenient for the subsequent comparison. A color equivalent to 0.005 g. of titanium in 100 cc. of solution will be found convenient. Water acidified with dilute sulphuric acid (equal parts acid and water) is then added to the standard tube to a volume allowing of subsequent necessary increase, 5 cc. of the hydrogen peroxide is added and the standard titanium solution added from a graduated burette in successive small additions until the colors agree. The measure of the standard solution required multiplied by its value in terms of titanium and corrected to agreement with the initial volume and weight is equivalent to per cent. of titanium. If the color of the solution yielded by a convenient weight of the sample is of insufficient depth for an accurate comparison, adjustment may be made as follows: A measured volume of the standard titanium solution, sufficient to furnish the required depth of color, is added to the solution being tested and the comparison conducted as above described. The volume of the standard solution required in the comparison minus the equivalent previously added to the test solution is calculated to the per cent. of titanium. The volume of the two solutions must, of course, be identical in the final comparison.

In preparing the standard titanium solution 1.05 g. of ignited c. p. titanic acid is fused with 10 g. of sodium carbon-

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ate. The melt is digested with 100 cc. of hot water to the complete solution of the soluble alkali, which is then filtered from the insoluble titanate. The residue of sodium titanate is washed with hot water containing a little sodium carbonate and dissolved from the filter with 100 cc. of dilute sulphuric acid (equal parts acid and water) and the solution diluted to 1 liter. The titanium equivalent of this solution is ascertained by igniting and weighing the precipitated titanic acid of an aliquot portion following a separation with sodium hyposulphite as previously described. The ignited salt will be found to contain approximately 95.6 per cent. of titanic acid; 1 cc. of the solution in the above proportion will therefore contain 0.6 mg. of titanium.

A gravimetric determination of the titanium in the iron may be conducted in the same manner, but inasmuch as in the colorimetric method the titanic acid as separated need be only relatively free from contamination by occluded salts, it is apparent that the latter method may be used to advantage, especially where many such determinations are required.

9. *Determination of Copper.*

A. Gravimetric Method.—5 g. of the sample is transferred to a glazed porcelain dish, 65 cc. of nitric acid (1.20 sp. gr.) is added, and the covered dish is heated until solution is practically complete; 25 cc. of dilute sulphuric acid (equal parts acid and water) is then added and the solution is evaporated to fumes of sulphuric anhydride; 10 cc. of the dilute sulphuric acid followed by 50 cc. of hot water is added, the residue is heated until in solution, and the solution is filtered into a No. 4 beaker and the filter washed with hot water. The filtrate is diluted to 300 cc., heated to boiling and, with the addition of 25 cc. of concentrated ammonium bisulphite solution, the boiling continued to the complete reduction of the iron; 30 cc. of a 20-per cent. solution of sodium hyposulphite is then added and the boiling is continued for 5 min. or until the precipitated copper sulphide has coagulated. The precipitate is collected upon an ashless 11-cm. filter, washed with dilute nitric acid (2 per cent.) and hot water, carefully ignited and weighed. The weight of the copper oxide as noted, multiplied by 16 (80 divided by 5) equals the per cent. of copper. Too great a tem-

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perature in the ignition causes the copper oxide to fuse on the crucible. As the copper oxide is somewhat hygroscopic, as are also some porcelain crucibles, less error is encountered by igniting in platinum.

The ignited copper oxide carries with it traces of contaminating iron. If it be desired to correct this usually negligible error, the impure oxide is dissolved in the crucible with 10 cc. of strong nitric acid, the solution transferred to a No. 2 beaker and diluted to 100 cc.; the iron is then removed by an ammonia precipitation. The precipitate so obtained is collected upon a 9-cm. filter, washed with hot water, ignited in the crucible in which the previous ignition was conducted and the weight of ferric oxide noted. This weight is deducted from the weight of the impure copper oxide and the correction applied.

B. Volumetric Method.—The nitric acid solution of the impure copper oxide in the No. 2 beaker, obtained as above described, is rapidly evaporated to the removal of the bulk of the free acid, and the solution diluted with cold water to volume of 150 cc. Strong ammonia is then added from a burette until the addition of a final drop causes the permanent bluing of the solution, after which there is added by means of a pipette six drops of acetic acid. 10 cc. of a 40-per cent. solution of potassium iodide is added, and the standard sodium hyposulphite solution is run in until the color of the liberated iodine has almost disappeared. 10 cc. of the starch solution is now added, and the addition of the standard sodium hyposulphite solution is continued to the final disappearance of the blue. The number of cubic centimeters of the standard solution required multiplied by its value in terms of copper and divided by five equals the per cent. of copper in the sample.

The value of the standard hypo solution is determined by titration of a solution of pure copper or by titrating the solution of a steel of known copper-content. The apparent large excess of the potassium iodide employed is because of its relation to the final end-point in the titration, the sharpness thereof decreasing in proportion as the excess of the iodide is decreased; the presence of ammonium salts in the solution also tends to lessen the permanency of the end-point, hence the necessity for the preliminary evaporation of the acid solution.

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If necessary, the time required for the volumetric determination may be materially lessened by initial solution of the sample in 50 cc. of dilute sulphuric acid (1 part acid to 3 parts of water), and the evaporation to dryness and filtration from the siliceous residue omitted. The iron in this case not being oxidized, the boiling with the ammonium bisulphite is also omitted. The copper oxide so obtained is contaminated to a greater degree by occluded iron, which if it be present in the final titration is prejudicial to a satisfactory end-point. This iron is, however, readily removed by filtration following the evaporation of the nitric acid solution previous to the addition of the ammonia. It is apparent, also, that following the nitric acid solution of the sample, the preliminary treatment with ammonium bisulphite in the gravimetric method may be dispensed with, as the reduction may be accomplished by the addition of a sufficient excess of the sodium hyposulphite solution at the time of the precipitation of the copper sulphide. The precipitate formed under such conditions, however, carries an unduly large burden of separated sulphur, the presence of which would appear undesirable.

SOLUTIONS REQUIRED.

Standard Sodium Hyposulphite.—8 g. to 1 liter; 1 cc. is equivalent to approximately 0.002 g. of copper.

Starch Solution.—The solution as used in the volumetric determination of sulphur.

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Recent Developments in Open-Hearth Steel-Practice.

BY N. E. MACCALLUM, PHŒNIXVILLE, PA.

(Cleveland Meeting, October, 1912.)

ALMOST half a century has passed since the Siemens brothers, after tedious and costly experiments, finally began the manufacture of open-hearth steel. The furnace of that time was very small, having a hearth-capacity of from 3 to 4 tons, and we can appreciate the wonderful development that has taken place in those 50 years by the knowledge that to-day furnaces of 50 times the original capacity are in operation. As the new steel process assumed importance, and furnaces began to grow in number and in size, a question arose, to which no satisfactory answer has ever been given, and that was: "What is the safe and economical limit of the open-hearth furnace?" Various stopping-places have been suggested, and 20, 30, 40, and 50 tons have each been advocated, but all such arbitrary restrictions have been passed, and 60 tons would probably represent the average furnace, although in a few instances from 70 to 80 tons are regularly tapped into one ladle. The furnaces which are the subject of the present paper have gone far beyond even these limits, and since, in my opinion, they mark another step in the progress of open-hearth steel, they will be considered fairly and frankly, in order that those to whom the matter may be of interest may be able to judge and form their own conclusions.

Short accounts of the work of these furnaces have appeared from time to time in various technical journals, of which the following is a brief *résumé*.

These furnaces are the property of the Phœnix Iron Co., of Phœnixville, Pa., and their distinctive and important feature is, that they charge to twice the capacity of one ladle, and tap simultaneously into two ladles, with a device for controlling the flow into either. In this way the capacity of the furnace is not limited, as formerly, but heats of any size within the capacity of two ladles may be charged.

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The ultimate reason for charging furnaces beyond the normal amount is not that large heats may be made, but because by making large heats a material increase of product is obtained.

Fig. 1 is a view of a furnace with divided spout, tapping into two ladles. A plan of the divided spout, pits, and ladles is given in Fig. 2. In order to make the branches of the spout as short as possible, one ladle is placed slightly in advance of the other, which also enables the trunnions to clear. Fig. 3 shows a plan and a front-elevation and Fig. 4 a side-elevation of the device for controlling the flow of steel into either ladle. Its purpose is not to stop, but merely to check, the flow. The levers by which it is operated are placed from 20 to 30 ft. from the center of the furnace, and the controlling-rod is protected by fire-brick, much the same as the ladle-rod.

The first of these furnaces, here designated as A, B, and C, was completed early in 1909. The area of the hearth, which had been decided upon after a few friendly visits to neighboring plants, was 32 by 14 ft., and was a fair average of furnaces charging from 50 to 55 tons of materials. At that time no thought had been entertained of breaking away from the established practice, but, on comparing the capacity of this hearth with those of our smaller furnaces, it proved to be so much in excess that the question was raised as to the advantage of a large hearth if it could not be fully utilized. Various experiments were then undertaken to determine the feasibility of using two ladles, and the best means of controlling the flow into either ladle, which resulted in our present practice.

Furnace B, completed in September, 1909, was the same in all respects as Furnace A, except that the hearth was lengthened 4 ft., making it 36 by 14 ft. in area.

Furnace C was built in 1911, the size of the hearth being increased to 42 by 15 ft. The size of the regenerator-chambers was also increased. These chambers were designed to meet existing conditions. The air-chambers of Furnaces A and B are 9 ft. wide and the gas-chambers 6 ft. wide. For Furnace C the air-chambers are 12 ft. wide and the gas-chambers 8 ft. wide. The length of chambers and depth of checker work are uniform for all the furnaces, 18 and 7 ft. respectively. If now these furnaces are rated as 115 tons, 130 tons, and 165 tons capacity, which is the usual weight of ingots tapped from each, it will be noted

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FIG. 1.—VIEW OF OPEN-HEARTH FURNACE WITH DIVIDED SPOUT, TAPPING INTO TWO LADLES.

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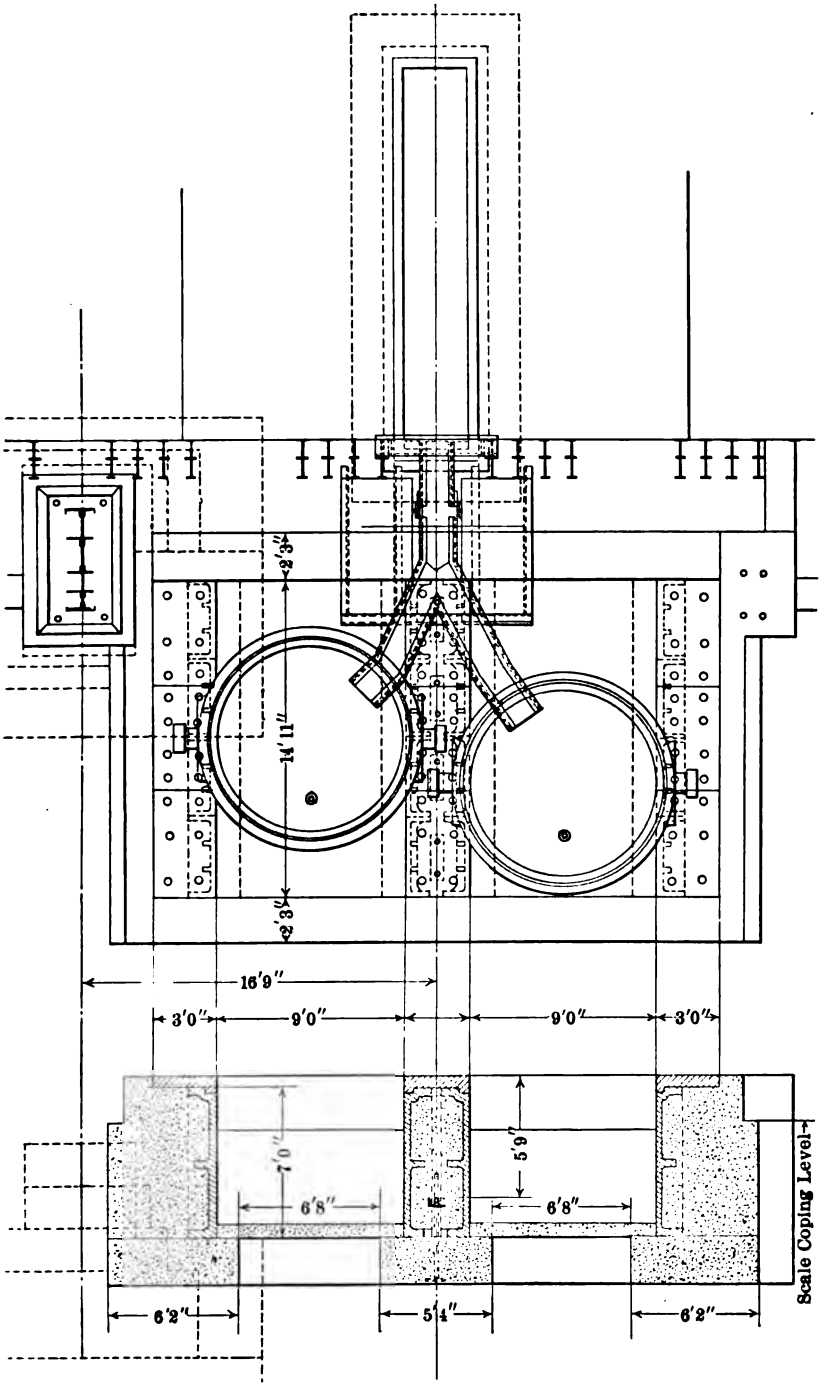

FIG. 2.—GENERAL PLAN OF TWIN LADLE-PIT AND SPOUT FOR OPEN-HEARTH FURNACES.

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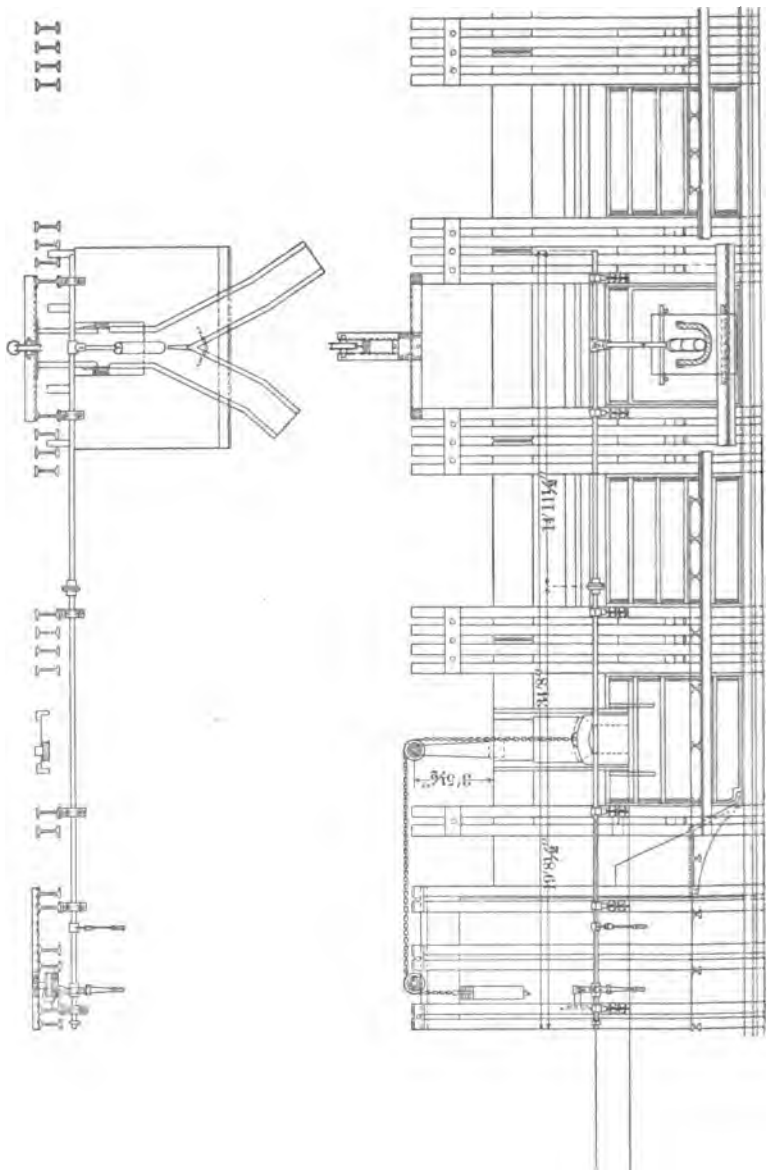

FIG. 3.—PLAN OF STREAM-CONTROLLER FOR TWIN RUNNER-SPOUT AND FRONT-ELEVATION OF OPEN-HEARTH FURNACE.

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that the checker-space on each end of the furnace is only equal to about 15 cu. ft. per ton of hearth-capacity; or, one-sixth of the amount generally allowed and recommended by prominent metallurgists. This statement is not made from any desire to criticise or detract from the prevailing views and practice; but it might be worth while to note that, had this practice been

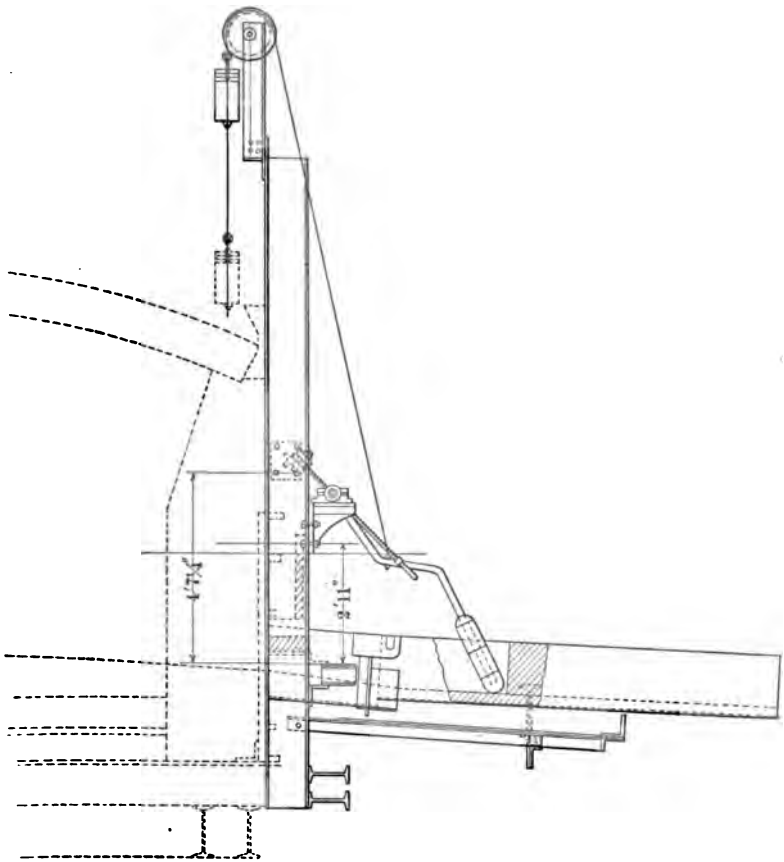

FIG. 4.—SIDE-ELEVATION OF STREAM-CONTROLLER.

followed, the depth of checker-space in Furnaces A, B, and C would have been 46 ft. instead of 7 ft. Or, to state the same thing in another way, the rated capacity of the furnaces, based on checker-space, would be 19 tons for Furnaces A and B, and 25 tons for Furnace C, instead of 115, 130, and 165 tons. It may therefore be safely assumed that any particular merit which

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this practice possesses must be attributed to causes other than the chambers.

The areas of the hearths of Furnaces A and B were practically the same as those of ordinary furnaces rated at from 50 to 55 tons capacity, and that of Furnace C, 42 by 15 ft., giving an area of 630 sq. ft., is almost an exact duplicate of the area of a number of furnaces recently completed and rated at 60 tons. The hearths are of the usual depth and measure from pan-plate to top of fore-plate 5 ft. 0.5 in. The pan is covered with 18 in. of brick and from 10 to 12 in. of magnesite in front of tap-hole. If, therefore, the hearths are of the same size, and the chambers are much smaller than those of many other furnaces, it is apparent that the large tonnage must be due to other reasons, which we shall endeavor to show lie not in the furnace, but in the practice.

On one occasion, in discussing methods of operation with the superintendent of a large steel concern, I asked if he reduced the size of open-hearth heats at the week-end, and the reply was that he did not, as it had been found that a large heat could be made almost as quickly as a small one. That was the open-hearth man's way of broadly stating a truth, which was not intended to be accepted too literally. It is not true, of course, that a large heat can be made as quickly as a small heat, but it is true that steel is made faster in a large unit than in a small one. This seems an obvious fact, yet it is one that receives but little attention and is rarely considered in comparing the work of one furnace with another. In most works, the number of heats which each furnace makes weekly constitutes the basis of comparison; but it is evident that for furnaces of different capacities it would be of little value, since the small furnace might make more heats and yet not equal the tonnage of a larger furnace, or, in furnaces of similar capacities, one might be delayed with repairs to hearth or otherwise and show a decreased tonnage, although it might be a faster-working furnace. For a number of years I have used, as a basis of comparison, the time from beginning the charging until the tap-hole is open, divided by the product, which gives, in minutes per ton, the rate at which steel is being made. This gives a reliable means of determining the speed with which any furnace is capable of making steel, and also its ability to compete with other furnaces. Using this

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method of comparison, Table I. has been compiled from furnace-records of the Phœnix Iron Co. In every case the record covers a period of not less than six months and should be fairly representative of the work done by each furnace. The smaller furnaces are of the older type with regenerators underneath the hearth, but did efficient work and compared well with other furnaces of similar construction. Furnace C now stands on the site formerly occupied by two of the smaller furnaces, Nos. 5 and 6, and it is interesting to note that Furnace C is making steel just a little more than three times as fast as No. 5 formerly did.

TABLE I.—*Data of Furnace-Practice.*

Furnace.	Hearth-Area.	Weight of Ingots.	Time Per Ton.
		Tons.	Minutes.
No. 3	17.5 by 12 ft.	30	32.79
No. 5	20 ft. 3.5 in. by 12.5 ft.	40	26.48
No. 1	24 ft. by 12.5 ft.	50	19.66
A	32 by 14 ft.	115	11.50
B	36 by 14 ft.	130	10.50
C	42 by 15 ft.	165	8.68

A study of Table I. will reveal two significant facts: One, that the economical limit, so far at least as time is concerned, has not been passed, since Furnace C is shown to be making steel 21 per cent. faster than Furnace B. The other, the striking decrease in minutes per ton as the hearth-area and unit of steel increase, and it is on this fact that the practice of the Phœnix Iron Co. is based. The extent to which any furnace may add to its product will be proportional to the excess charged over the normal one-ladle heat, but from the experience of the Phœnix Iron Co., it may be affirmed that by doubling the normal charge the product of any furnace can be increased by from one-fourth to one-third; that is, if a furnace charging single-ladle heats can produce, say, 600 tons weekly, it will by charging double-ladle heats produce from 750 to 800 tons. This can be readily demonstrated by operating any furnace alternately on single-ladle heats and half-ladle heats, and comparing the number of minutes required in each case to make a ton of steel.

Why should steel be made faster in large units than in small? No particular reason can be assigned, but there are several

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probable causes which contribute to that result. The first and most important is the economy in time; that is, in a given time more steel will be produced, or the same amount of steel will be produced in less time. This can be better understood by dividing into two periods the time required to make a heat of steel, the first period being the time required to charge and melt, and the second period the time from when the lime floats to the surface of the bath until the heat is ready to be tapped, known as the "working" of the heat. In the first period the double-ladle heat will not show much gain over the single-ladle heat, as it will take almost twice as long to charge and melt, but in the second period, when spar, ore, and other additions are made, the double-ladle heat gains a decided advantage, since it takes but little longer to "work" a large heat than a small one. At various times during the past three years single-ladle heats have occasionally been made on Furnaces A, B, and C, and a careful scrutiny of the records shows that the average time required to make a ton of steel on those heats was 12.2 min., while the average time at which furnaces were running on double-ladle heats was 9.7 min., showing that it required 25.7 per cent. more time to make a ton of steel on a single-ladle heat than on a double-ladle heat.

As a second cause, it is evident that twice as many single-ladle heats will be required to produce the same tonnage as that from double-ladle heats, therefore one-half the time which would be required to make repairs to the hearth in the former case will be available for making steel in the latter. Then again, each heat of steel that is made, whether large or small, entails two extremes of temperature, viz.: the cooling of the hearth and furnace when cold stock is introduced, and the maximum temperature necessary to tap, at which time the hearth and furnace must be raised to at least the same temperature as the steel. It will therefore take twice as many heat-units to raise the furnace and hearth to this temperature to make two single-ladle heats as would be required to make one double-ladle heat, just as in heating water or other substance in a refractory vessel, less time will be required to heat a certain amount in one portion than to divide that amount into two portions and heat each separately, the difference in time being that required to heat the container a second time.

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A third cause is radiation. Campbell,[1] in reference to heat-distribution in an open-hearth furnace, says: "Roughly speaking, about one-half of all the heat supplied to an open-hearth furnace is lost by radiation and conduction." That statement concerns the present paper only to the extent that the more frequently a furnace is raised to a tapping-temperature the greater will be the loss by radiation, as the latter increases with each degree of raise of temperature, so that the fewer heats that are made in proportion to the tonnage, the less frequently will the furnace be raised to a tapping-temperature, and the less will be the loss by radiation. This is simply illustrated by noting the time required to heat a volume of water from t° to the boiling-point, then removing a portion and replacing with water at t°, and again raising to the boiling-point, and repeating the operation until the added portions equal the original volume. The added portions collectively will take longer to boil than the original amount, the difference being equal to the excess radiation that has occurred at the higher average temperature.

These views have been advanced with some hesitancy, but they appear to offer a reasonable explanation why steel is made faster in large units than in small ones. To confirm these arguments further, a statement of the actual tonnage which each furnace has made is given below. All the stock is charged cold, and the fuel used is producer-gas.

Furnace.	Period of Time.	Output.	Time Per Ton of Output.	Fastest Time for One Ton of Output.	Largest Heat.
		Gross Tons.	Minutes.	Minutes.	Gross Tons.
A,	First year.....	41,436	11.45	8.27	130.04
	Second year.....	39,910	11.54		
B,	First year.....	43,900	10.38		
	Second year.....	43,615	10.44	7.49	146.32
C,	47 weeks.....	47,200	8.68	6.92	177.77

Furnace C has still five weeks in which to complete the year and should run well over 50,000 tons, which is probably the largest tonnage that has ever been produced from any furnace using all cold raw material.

The saving in cost, which will be proportionately increased as the tonnage increases, depends upon labor, fuel, and repairs.

[1] *Manufacture and Properties of Iron and Steel*, 3d ed., p. 224 (1904).

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The number of men necessary to operate a furnace on a double-ladle heat is the same as on a single-ladle heat, and if the product is increased one-third, it is apparent that the labor will be reduced one-fourth, including all furnace-labor except that of the stock-loaders.

The fuel (gas-coal), covering a period of several months, will not vary much whether the furnace is making large or small heats, and, as is the case with the labor, should show a saving of one-fourth. Furnaces A and B are on the same gas-line as the blooming-mill soaking-pits, and, consequently, it is difficult to get accurate data. Furnace C, however, is supplied from an independent line, and the figures used are based on the actual amount of coal received in the cars. On a run of 22 consecutive weeks, including the entire time from commencing the first heat until shut down for repairs, the average amount of coal used per ton of steel was 547 lb. Just whether this represents a saving of one-fourth is uncertain, but we can estimate what the amount of coal would have been had the furnace been operated on single-ladle heats, in which case the product would have been reduced one-fourth and the amount of coal increased one-third. Furnace C averages slightly more than 1,000 tons of product weekly, so the product would have been 750 tons, and the coal increased one-third would have been 729 lb. per ton of steel. That is, instead of 1,000 tons of steel, with a coal-consumption of 547 lb., on double-ladle heats, we should have 750 tons of steel, with coal-consumption of 729 lb., on single-ladle heats.

Bearing in mind the arguments that were advanced in explanation of the increased product of large heats, the use of pig-iron with an average content of 0.70 phosphorus and 0.70 silicon, with considerable quantities of lower-grade iron as cheapeners, together with the miscellaneous scrap that the market affords, 729 lb. of coal cannot be considered excessive for Eastern practice, particularly covering a period of 22 weeks, so that it is probable a saving of one-fourth is not far wide of the mark. With larger regenerators it is likely a still further saving might be effected.

What has been said in regard to labor and fuel should also apply to repairs. That is, in a given period the wear and tear will be about the same, regardless of the size of the heat or of

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product. In the run of 22 weeks, the total product from Furnace C was 24,298 gross tons, and the repairs that followed, including roof, linings, checkers, cost of brick, bricklaying, and common labor, was a fraction less than 12 cents per ton.

Having discussed the increased product and economies resulting from the two-ladle practice, what are the objections likely to be raised against this method of steel-making? Probably the first would be the risk of handling such large quantities of steel and the possible consequences in case of accident. In reply, it may be said that in modern open-hearth practice, when linings have been increased in thickness, partly to overcome excessive radiation and partly to prevent accidents, break-outs rarely occur, and as double-ladle practice requires only one-half the usual number of heats to produce the same tonnage, it is evident that that risk, whatever it is, has been cut in half. It may be imagined that serious trouble might result from the chilling of a heat in the furnace, as occasionally happens when the roof falls in. I confess that the first instance of this kind was viewed with some apprehension. It occurred on Furnace B, when a charge of 335,000 lb. was melted; the roof from hip to hip fell in. This happened at 8 a.m. on a Saturday, and during the next day the roof and lining were replaced. Gas was turned in at 5.30 a.m. Monday, and the heat was tapped at 10 p.m. Tuesday, yielding 139 gross tons of ingots. A similar experience occurred another time, but in neither case were any difficulties presented other than would be encountered in smaller heats. The divided spout and controller have at all times worked satisfactorily, and no trouble has been experienced in dividing the steel, although occasionally one ladle may receive more than the other, but generally both ladles are evenly filled.

Another objection that might be raised is, the difficulty the mills may have in taking care of the product in large units, but a little consideration will show that when mills are provided with sufficient soaking-pit capacity, it makes but little difference. If the product is supplied from large furnaces, there will be fewer heats, whereas, if supplied from smaller furnaces, there must, necessarily, be more furnaces, and a largely increased number of heats, frequently causing what mill-men call "bunching" of the heats, that is, when several tap together or in close

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succession. That the large heats are not objectionable to the mill-men is evidenced by the interest which they show in the tapping of the large furnaces.

As to the quality of the steel, it may be said very positively that there is no apparent difference from that made in smaller amounts. The chemical analyses, the physical tests, and fabrication of hundreds of thousands of tons, would surely have detected any difference had such existed.

I have traced, within the limits of this brief paper, the causes which led to the development of the double-ladle practice, and have shown the increased tonnage and decreased cost resulting therefrom, also giving the objections that may be urged against this practice. In conclusion, I may say that my object is, not to present the Phœnixville furnaces as types of fast-working furnaces, as they are not so considered, but to show by the work that they have done what can be accomplished by the practice which they exemplify

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Methods of Preparing Basic Open-Hearth Steel for Castings.

BY H. F. MILLER, JR., VERONA, PA.

(Cleveland Meeting, October, 1912.)

FOR some years the prejudice against basic open-hearth steel for casting has been gradually decreasing. Yet many consumers and engineers still cling to acid steel for castings, because of their alleged greater freedom from blow-holes and sponginess.

Acid steel has been used for this purpose much longer than basic steel; and the melters in acid practice had it well in hand when basic steel was first tried. Then the necessity of learning a new set of laws for the production of satisfactory basic open-hearth steel for castings became evident.

The first of these laws, in my opinion, has to do with the furnace construction. The heat should be melted down as speedily as possible so as to prevent excessive oxidation. I have described in another article[1] some furnaces designed by me, and having as one of their features the promotion of quick melting.

Size of the Furnace.

The hearth of the furnace, however, is a decisive factor in the production of solid castings. The manufacturer should know the size of the heats he intends to make constantly, and should have his furnace built for that size of heat. The hearth should differ in dimensions from that of a furnace making ingot-steel. That is, the bath should be deeper and should have less surface-area. A shallow bath permits the slag to come out soon after the steel commences to flow, and thus prevents the additions from going into the steel, or from becoming uniformly distributed in case they have been put in hurriedly.

Under this head comes the very poor practice of making small heats in hearths of a much larger capacity. If into a 25- or 30-ton furnace only 12 or 15 tons of metal is charged per heat, the proportion of heats that will be wild, or show signs

[1] *Iron Age*, vol. lxxxix., No. 26, pp. 1595 to 1596 (June 27, 1912).

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of wildness at some time during the pouring, will be comparatively large; whereas, when the hearth is charged to capacity, a heat showing signs of wildness will be a rare occurrence.

Influence of the Slag.

The nature and action of the slag is an important factor in the manufacture of quiet steel. Slags are usually roughly classified by the melter according to physical appearance, as follows:

1. The "dry," heavy slag occurring when there is very little silica present. This is a dangerous slag if not carefully worked. The burning of many furnaces is due to reflection of the heat to the roof by this slag. Another danger is, that the melter, deceived by the physical appearance of the slag, may add an excessive amount of fluorspar. This results in a badly-cut ladle and a stopper-rod burned off. These disasters can be prevented by a gradual addition of spar until a wet slag is created, after which the heat may be worked down as usual. With natural gas the heat will foam for some time.

2. A "wet" but "lumpy" slag. This is a good slag to work with. The lumps of limestone should be broken up with a rod, so that a rocky bottom may be avoided. In some cases an unbroken lump will choke the tap-hole, so that, the flow of the slag being stopped, the steel is left uncovered until the tap-hole can be freed. A large amount of heat is lost from the steel thereby. A lumpy slag can be avoided by charging small size limestone.

3. A third slag is the very watery variety, usually occurring when heats melt high in carbon, by reason of the presence of an excessive amount of silica. This slag should have burned limestone, burned dolomite, or raw limestone added until a thick slag is made. When the slag is too thin, it will mix with the steel in tapping and a wild steel will be the product. The ladle and stopper-rod will be badly scorified, and usually some tons of steel will go into the pit by reason of a burned-off stopper-rod.

The ideal slag is heavy and wet, with no large lumps. This slag makes an easy heat to work and gives a steel low in phosphorus and sulphur. It requires but a small amount of fluorspar to put this slag in shape for the ladle. A heavy slag of

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this nature will not mix with the steel, and will generally stay in the furnace until the steel is nearly all in the ladle. It also has the good quality of cutting neither the stopper-rod nor the ladle-brick. The only objection is that it causes a dirty bottom, and unless a "washout" is made after each heat, the bottom will rise to the sill-plate level after a few heats. This result shows very poor practice. No time is ultimately gained by charging up without the usual "washout."

If a "washout," requiring from 20 to 30 min., is made after each heat, the furnace will work fast for a much longer time. Moreover, the holes that occur in a "high" bottom will be largely avoided, if the bottom is kept low and clean.

The Tap-Hole and Spout.

The tap-hole should be kept large and low. The quicker steel gets into the ladle, the more heat is retained. The shorter a spout, the better, for the same reason. Moreover, a large tap-hole will not clog up easily if rabbling has to be done; and few "hard" taps will occur. In shutting up the tap-hole magnesite is best, because it does not burn together, making a "hard" tap, nor does magnesite boil out, as may happen when a tap-hole is closed with green dolomite. Burned dolomite, however, may be safely used, the only drawback being an occasional hard tap, due to the burning together of the material.

The above facts are probably known to most melters; but, unless vigilance is constant, one factor is apt to be overlooked. The bottom may be low, and the tap-hole all right; but the spout if not smooth at the end may cause the stream to spray over the ladle, thus losing much heat and causing a skull and, perhaps, some misrun castings, all of which could have been avoided if the spout had been made up carefully.

The Addition of Alloys.

The addition of alloys may be made in the bath or in the ladle. By putting them into the bath, much heat is saved. This is a valuable method where a furnace is working cold or a heat has melted "low" and there is difficulty in getting it hot. The advantage is that the additions are made while the flame is still on the bath, and the loss of the heat in the bath caused by dissolving the alloys can be regained. The objections are: (1) that a large amount of each alloy must be added,

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since, in this way, from 15 to 30 per cent. goes into the slag; (2) that the silicon, reacting, will throw back the phosphorus into the steel.

Putting the additions into the steel as it goes into the ladle is, in my opinion, the better method. A uniform distribution of alloys is attained by shoveling the alloys in gradually.

If the alloys are added in the ladle, the steel will be helped greatly by first raising them to red heat, especially in winter.

The Working of the Steel.

After the heat has melted down and the limestone has boiled up, the charge will be benefited by allowing it to "soak" from 15 to 30 min. This will allow any contained slag to rise, and also much of the gases.

Ore may then be fed if necessary, and, after it has done its work or after all of the limestone has boiled up, the heat should be soaked for an hour or more before tapping, so that much of the occluded gases may escape. It is my opinion that brittle steel is caused not only by phosphorus, but by the gases in the steel as well, since I have noticed steel low in phosphorus to be brittle, and steel of like analysis after soaking for an hour or more to be very tough.

The carbon should be "caught coming down" rather than by recarburizing in the ladle, because there is danger of forming gases which will cause wild steel. This is the case if anthracite coal is used for recarburizing. Sometimes, if a heat melts low, it is necessary to add carbon to the steel, and I have found that it is best to recarburize with carbon of the same nature as that in the steel, such as powdered petroleum-coke, lampblack, etc., for the steel rarely shows wildness when using carbon of that nature.

Sometimes heats will melt low, and if one be hampered with small ladles, or have no heavy castings to pour, it is best to get the heat ready by using manganese. This will not add much metal to the bath, and it is a good substitute for pig-iron.

With large ladles, I would recommend working the heats with both pig-iron and manganese. Very little, if any, ferro-silicon should be used instead of manganese, since the silicon mixes with the slag and cuts off the stopper-rod while the heat is being poured.

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"Hot metal" is preferable to cold additions, as the bath is not then chilled by the addition.

The great danger in working "stickers" is in the treatment of the slag, which often looks thick and heavy by reason of a cold bath. The mistake is made constantly of adding too much fluorspar. The result is that when the bath begins to pick up heat "the slag turns to water." If the charge is tapped with the slag in this condition, and it takes more than 30 min. to pour, the steel remaining in the ladle will be lost, as the stopper-rod will be burnt off and the ladle badly cut. On the other hand, if the slag is watery and the attempt is made to thicken it, much heat will be lost, and time consumed before it will be ready to tap. This suggests the good rule "Add fluorspar sparingly," whereby expensive mistakes will be avoided.

If the slag comes too quickly and all the additions have to be made to the first part of the heat, an even distribution can be obtained by rabbling the heat. If some of the additions are lost in the slag, or if not enough have been put in, the heat showing signs of "wildness," a simple remedy is to take a number of sticks of aluminum, bend a tapping-rod around them, thrust them to the bottom of the ladle, stir the bath with them, and rabble it afterwards if necessary. There is hardly a heat that cannot be made absolutely quiet by this means. But it is a curative treatment, and a good preventive is always better.

The Time of Pouring.

The time consumed in pouring is a factor in determining the size of the heat. Good practice demands that a heat of steel should be poured in less than 60 min. The faster the better. To take longer than that results in a reaction between the steel and the slag, which causes: (1) a loss of silicon and a gain in phosphorus and sulphur in the steel; (2) the exhibition of "wildness" in the steel; (3) the skulling of the ladle; (4) misrun castings; (5) high ladle-cost due to the cutting of the brick by the slag; and (6) overheating the stopper-rod, the bending of which causes the loss of the remaining steel.

Again, the higher temperature required to pour steel from 1 to 2 hr. causes a great increase in occluded gases and necessitates an increase of about 30 per cent. of ferro-silicon to make it as quiet as when poured at lower temperature.

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The use of fluorspar seems also to make the steel less responsive to the quieting action of silicon, since the fluorine seems to be absorbed by the steel. I have noticed when pouring a test that after much spar has been used, the steel gives off a smoky gas of the same appearance as when fluorspar is added to the bath. If this is true, the action of fluorspar cannot be beneficial to steel when added in large quantities.

Good basic steel is harder to make than good acid steel. But if the above outline is followed the steel produced will be as quiet while molten and as solid afterwards as acid open-hearth steel—and tougher.

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Recent Developments in the Inspection of Steel Rails.

BY ROBERT W. HUNT, CHICAGO, ILL.

(Cleveland Meeting, October, 1912.)

PERHAPS of all the scientific economic questions which have been claiming the attention of capitalists, metallurgists, manufacturers, directors of public utilities, and the general public of America, during the last few years, none have been of more importance than the one relating to the wear and safety of steel rails.

The discovery of the Bessemer process made possible modern railway development. In that growth, as has generally happened in all progressive movements, all its departments have not remained equally balanced. Iron rails permitted the construction of roadways for the passage of power-driven vehicles. In time the speed and weight of such carriages proved too much for the wearing-qualities of the rails, and the limit of railway capacity seemed to have been reached. Bessemer made his invention, which was soon perfected and developed, especially as related to steel rails, and thereby increased the limit of capacity, and vastly greater railway progress was made possible. The Siemens-Martin open-hearth process followed, and was later supplemented by the Thomas-Gilchrist basic improvement.

For a time the Bessemer steel rails of commerce were more than equal to all that was asked of them, but this did not long continue, as evidenced by the records in our *Transactions*. Revolutionary changes occurred in the industrial world, and in no part of it more than in steel-rail manufacture. Great and powerful commercial combinations were made, whose financial resources made possible the supplanting of the then most efficient blast-furnaces, steel-works, and mills, by others of theretofore unimagined capacity. Old traditions and practices of steel-making were wiped away, and, in too many cases, output was apparently the chief aim, with the result that much dissatisfaction with the wear and safety of rails was felt by railway officials, and the general public was concerned as to its safety when traveling. The climatic conditions of the winter

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of 1911–12, extending over practically the whole of North America, were abnormally severe. A number of accidents occurred through which there was intense suffering and appalling loss of human life. Government investigation threatened to be followed by government control.

The question of chemical and physical specifications governing the manufacture of rails, and the form and weight of the sections, had already been given, and continued to receive, much attention from committees of scientific societies, individual engineers, and metallurgists. There was much difference of opinion as to the proper corrective action, but a consensus of opinion that something should be done, and that quickly. Several conferences between representative railway and steel company officials were held, during which it was made manifest that the desire existed on both sides to treat the question fairly and make honest efforts to reach the best solution of the difficulties. There were millions of dollars involved, but far beyond that came the protection of human life; and something more than conference resolutions was required.

It had long been the general custom for railways to have their rails inspected by their own representatives before their delivery to them by the manufacturers. The extent of this inspection varied with different roads, but it was generally limited to an examination of the rails after they were finished and offered for delivery. This embraced inspection as to accuracy of section, of length, squareness of sawing, correctness of drilling, straightness in line and surface, and freedom from visible physical defects. Most rail contracts, through specifications, provided for chemical composition, physical treatment of the ingots, a limitation of finishing-temperature, and physical testing of representative pieces of the rails; but the accurate carrying out of such details was largely left to the employees of the manufacturers, although practically all contracts specified that inspectors representing the purchaser should have free access to the works of the seller while the rails were being manufactured.

The foregoing outlines the situation as it was in the early part of 1912, when the chief executive officers of several of the principal American railway systems decided to endeavor to obtain better results under their current specifications and rail-sections; or at least endeavor to secure accurate data upon which to determine the actual cause of so many rail-failures.

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They reasoned that, as the majority of their rails did not break in service, and gave reasonably satisfactory wear, it would seem that the failures of the lesser percentage could not be entirely due to faultily proportioned or too light sections, poor road-bed, too heavy wheel-loads, or excessive speed of traffic. They, therefore, arranged to try a more thorough system of inspection, by placing inspectors in each manufacturing department of the rail-plants, to be on duty during all working-hours. It was realized that the manufacturers could not be expected to permit the inspectors to dictate how the details of operation should be conducted, or to interfere with the men in their work; but they were aware that the workmen were paid for work on a tonnage basis, which, of course, had the effect, as it was intended it should, of making each and every man anxious to produce all the output possible, and, moreover, that there was not any effectual way of connecting the quality of the labor but of a few of the workmen with individual rails. Therefore, the instructions to the inspectors were to observe carefully all of the manufacturing operations in the department of the works to which they were assigned, and, if they saw any serious variation from the details of the specifications under which the rails were contracted to be made, or from the recognized good practice of that particular plant, they were at once to call the attention of the foreman in charge of that department to what was happening, make a record of the fact, and also report it to the inspector in the succeeding department to which the ingot, bloom, or rail would pass in the progress of its making. This knowledge would cause extra care on the part of the inspectors in the observation of the behavior of the metal in the rolls, at the drop-test, and the most careful examination of the finished rails for defects, assuming that the heat of metal from which they were made had successfully passed the prescribed physical tests; and another important point was to be certain that the rails were correctly marked as to their heat-numbers and respective locations in the ingots from which they were rolled, and that the pieces of rail for drop-testing were accurately taken. The particulars of the inspectors' observations were to be reported in writing to the proper officer of the purchaser, and in any case of neglect of duty or bad work, the fact was to be given in writing to the official in charge of the works, thereby affording an opportunity for immediate investigation, and so be a

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check on the judgment and fairness of the inspector making the report.

It was believed that the moral effect on the workmen of knowing that their individual responsibility and association with a heat of steel, and the rails made from it, did not cease with its passage from the department in which they were employed, or being shipped from the works, would tend towards their more careful, and, therefore, better, work.

The engineering staff of all large railway systems keep records of the results obtained from the rails used, and if possessing such data as would come from the foregoing described plan, it would be possible to trace the creative history of rails, should they have failed, or, better still, given satisfactory service.

When the plan of the greater inspection was presented to the rail-makers, they, with scarcely an exception, accepted it cheerfully, and why not? They desired to produce good rails, and wanted to do so with profit to their stock-holders. They paid their workmen for faithful and intelligent service, and, in the long run, it is always more economical to do things right than wrong. By the placing of extra inspectors in their works, they were given that many additional monitors over their men without cost to their companies.

The plan calls for placing men on day- and night-service in the steel-producing departments; in the blooming-mill; in the rail-mill, and in charge of the drop-testing; also from 4 to 5 men on the finished rails; making a total of from 12 to 15 inpectors at each plant. This inspection is now used by 33 railway systems, operating more than 122,000 miles of track, and up to Oct. 1,1912, more than 650,000 tons of rails have been shipped. The plan is working with but little friction, and is proving so satisfactory that it will undoubtedly be adopted by other railway systems. It places the companies using the plan in the strong position of using their best efforts to protect the traveling public.

Naturally, the inauguration of so much detail and the assembling of so large a corps of men, as each and all of the 15 rail-mills of the United States and Canada had to be covered, involved much labor and some change in the personnel of the corps, and alterations of and additions to the forms used in making the reports. I append as a matter of information and record a complete set of the forms now being used in the work.

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ROBERT W. HUNT & Co.
1121 Rookery, Chicago, Ill.

SPECIAL INSPECTION REPORT RAIL MILL. (Size of original form, 5½ by 8½ in.)

From _____ Mill. between _____ Date and Hour. to _____ Date and Hour. Report No. _____

Rails for _____ Railroad. _____ Inspector.

No. blooms per ingot _____ Size _____ No. rails per bloom _____ No. passes in blooming mill. _____

Heat No.	Time. Charged.	Time. Rolled.	No. Ingots Bloomed.	Remarks on Heating and Blooming.	Per Cent. Discard. Top.	Per Cent. Discard. Bottom.	Time Charged to Reheat Furnace.	Remarks.
1 (15 lines, each numbered consecutively.)								
2								

Use other side for full details. Under "Remarks" show heats reported to rail mill inspectors. Send this report to general office promptly.

ROBERT W. HUNT & Co.
1121 Rookery, Chicago, Ill.

SPECIAL INSPECTION REPORT RAIL MILL. (Size of original form, 5½ by 8½ in.)

From _____ Mill. between _____ Date and Hour. to _____ Date and Hour. Report No. _____

Rails for _____ Railroad. _____ Inspector.

Section _____

Use other side for full details—Send this report to general office promptly.

Heat No.	Time Rolled.	No. Rails Rolled.	Template Fit.	Condition of Section.	Maximum Shrinkage.	Per Cent. Discard. Top.	Per Cent. Discard. Bottom.	Condition of Sawing, Stamping, Lettering, Branding.	Hot Trial Weight.	Condition Rails Leaving Hot Beds.
1 (15 lines, each numbered consecutively.)										
2										

FIG. 1.—FORMS FOR INSPECTION OF BESSEMER STEEL.

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ROBERT W. HUNT & Co.
1121 Rookery, Chicago, Ill.

SPECIAL INSPECTION REPORT BESSEMER STEEL. (Size of original form 8½ by 11 in.)

From _____ Between _____ To _____ Report No. _____
Mill. Date and hour. Date and hour.

Rail steel for _____: _____ _____
Railroad. Inspector.

Give full details on other side.

Report only Heats made on this Order—send this Report to General Office Only.

Average time blowing heats _____ No. of converters used _____ size of nozzle _____

Was spiegel added to vessel or ladle _____ Other additions made to ladle _____

Average time ingots held in molds before stripping _____ Additions to molds when teeming _____

Treatment of ingot tops _____ Treatment of molds and stools _____

Size of ingots _____ Nominal No. ingots per heat _____ Average ingot weight heats cast _____

Heat No.	Condition and Temperature of Blowing.	Additions to Metal in Vessel During Blow.	Time Metal Held.			Reaction after Adding Spiegel.	Condition of Teeming.	Condition of Ingots.	Remarks.
			In Vessel after Recarbonizing.	In Intermediate Ladle.	In Teeming Ladle before Pouring.				
1									
2 (30 lines, each numbered consecutively.)									
3									
4									

FIG. 2.—FORM FOR INSPECTION OF BESSEMER STEEL.

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ROBERT W. HUNT & Co.
1121 Rookery, Chicago.

SPECIAL INSPECTION REPORT OPEN HEARTH STEEL. (Size of original form 7¾ by 8½ in.)

From _____ Mill. Date _____ Time worked. Heat No. _____

Steel for _____ Railroad. Inspector.

Additions to furnace _____ Condition bath, slag, reaction _____

Additions to ladle _____ Reaction and temperature _____

Time tapped _____ Weight ingots cast _____ Size nozzle _____

Time furnace empty (ladle full) _____ Size of ingots cast _____ Additions to molds _____

Ingot No.	Conditions of			Time.		Remarks.	Ingot No.	Conditions of			Time.		Remarks.
	Molds and Stools.	Teeming.	Ingots.	Cast.	Stripped.			Molds and Stools.	Teeming.	Ingots.	Cast.	Stripped.	
1	(20 lines, each numbered, and space for remarks.)							(20 lines, each numbered, and space for remarks.)					
2													

Send this report promptly to General Office.

ROBERT W. HUNT & Co.
1121 Rookery, Chicago.

SPECIAL INSPECTION REPORT BLOOMING MILL. (Size of original form 7¾ by 8½ in.)

From _____ Mill. Date _____ Time worked. Heat No. _____

Size of blooms _____ No. of rails per Bloom _____ Inspector.

No. passes in blooming mill _____ Inspector.

Ingot No.	Time Charged to Pit.	Charged to		Conditions of Heating.	Time Bloomed.	Condition of Blooming.	No. Blooms Made.	Per Cent Discard.		Condition of Bloom after Shearing.	Time Charged to Reheat Furnace.	Ingot No.	Time Charged to Pit.	Charged to		Conditions of Heating.	Time Bloomed.	Condition of Blooming.	No. Blooms Made.	Per Cent. Discard.		Condition of Bloom after Shearing.	Time Charged to Reheat Furnace.
		Pit.	Hole.					Top.	Bottom.					Pit.	Hole.					Top.	Bottom.		
1		(20 lines, numbered for space marks.)		each, and re-												(20 lines, numbered for space marks.)		each, and re-					
2																							

REMARKS.

Send this report promptly to General Office.

FIG. 3.—FORM FOR INSPECTION OF OPEN-HEARTH STEEL.

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(NOTE.—The heading of the original form, here given for convenience in three sections on this and the opposite page, extends across the top of the form, which is 11 in. high and 22 in. wide. Below this heading 30 horizontal lines are ruled, each bearing a serial number.)

ROBERT W. HUNT & Co.,

1121 Rookery, Chicago, Ill.,

SPECIAL INSPECTION REPORT OF STEEL RAIL MANUFACTURE.

For _____ Supplementing our rail report No. _____

Rails rolled by _____ This report No. _____

_____ Steel rails _____ Lbs. per yd. _____ Section _____ Date rolled _____

Section 1.

	Heat No.	Date Made.	Size of Heat Cast.	Additions Made to Metal in Ladle	Time Ladle Held Before Casting 1st Ingot.	Additions Made to Metal in Molds.
1						
2						

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Section 2.

Size of nozzle used for casting ingots_____ Exceptions_____

Size of ingots cast _____ Approximate average weight_____

Size of blooms_____ No. blooms per ingot_____ No. rails per bloom _____

No. passes given ingots in blooming mill_____ Exceptions_____

No. passes given in rail mill _____ Total No. passes ingot to rail _____

Remarks on Ingot Molds and Stools, Casting Ingots, Ingots, Top and General Conditions.	No. of Ingots Cast.	No. of Butts and Stickers.	Time Between Casting and Stripping Ingots.	Time Between Stripping and Charging Ingots to Pits.	Time Ingots in Pits before Rolling.	No. of Ingots Bloomed.

Remarks.

Section 3.

Remarks on Heating and Blooming.	Approximate Per Cent. Discard.				Time Blooms in Reheating Furnace	No. Rails Rolled.		
	At Shears.		At Saws					
	Top.	Bottom.	Top	Bottom.				
								1
								2

FIG. 4.—FORM FOR INSPECTION OF STEEL-RAIL MANUFACTURE.

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Discussion.

H. D. Hibbard, Plainfield, N. J.:—What is the usual amount of discard from the top of the ingot? Does it often happen that additional discard is made in order to secure soundness of the top rail of the ingot?

Mr. Hunt:—There is no uniform practice. The generally accepted one, without its being written in the contract, is that the discard shall amount to about 9 per cent. Many of the specifications provide that it shall continue until seemingly sound steel is reached in the bloom. A few, but very few, contracts have a specific discard of from 20 to 25 per cent., but that is exceptional, and for that an additional charge is made by the manufacturer. That is one of the questions which is now agitating the purchasers of rails—that is, the railway people—as to what future steps they shall take. There is a very strong tendency towards insisting upon at least one-third of the ingot not being used for rails. One railway company purchased a large quantity of rails where, without its being so stated in the contract, it was understood that the top bloom would not be given to them. The manufacturers, fortunately for them, had two rail-mills, and they diverted the top bloom to the other mill, where it was rolled into rails of other sections. Who got them I am not prepared to say.

Wm. H. Blauvelt, Syracuse, N. Y:—I should like to ask Mr. Hunt whether the same organized effort is being made to improve the uniformity of the steel in the ingot before it passes to the rolls, among the steel-makers as a whole, as he has described in this treatment of the steel by the rolling-process.

Mr. Hunt:—This form of inspection covers the metal from the time it is tapped from the furnace or poured from the converter; its action in the ladle; what additions are made to it, and if there is any variation from the works' accepted good practice, as, if the slag comes ahead of the steel in tapping; if the ladle-stopper does not work well, and it is a badly-poured heat; if cracked ingot-molds are used; if some of the ingots are stickers, etc.—in fact, the whole history of the process becomes a matter of record, without any interference on the part

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of the inspector. The result has been, I think I am quite safe in saying, that there have been a great number of heats (I will not try to give accurate figures) that have not been rolled into rails, which, under old conditions, would have gone. They have been diverted to other purposes, or put aside, and not applied to that particular rail-contract—and I say this without any disrespect to the steel-makers, I mean the managers of works. No manager can be in all parts of his works at all times and see everything, and as long as the men are paid by the ton, it is human nature to try to get the product past.

FREDERICK LAIST, Anaconda, Mont.:—I read somewhere that about 2 per cent. of copper was occasionally added to rail-steel for the purpose of toughening it. Has this been done, and, if so, has it given successful results?

MR. HUNT:—The Chicago, Milwaukee & St. Paul railroad had 5,000 tons of rails made from steel to which copper was added; it was expected to secure about 0.5 per cent. of copper in the rail; that is, they were seeking to obtain about 0.5 per cent. of copper in the metal. These rails were made by the Illinois Steel Co. The manufacturing results were entirely satisfactory. The copper was added to the steel in the casting-ladle in the form of copper wire, so as to get small individual pieces. The idea originated with the railway; in fact, I might say that it was due to E. W. McKenna, Vice-President of the Chicago, Milwaukee & St. Paul railroad, who has always taken a great interest in steel rails and was the inventor and introducer of the McKenna process, so called, for re-rolling worn rails of large sections and still keeping them to standard sections, which practice has been successfully working now for some 12 or 15 years. Mr. McKenna's idea was based upon the satisfactory results which had been obtained from the rails made by the Bessemer process from ores from Cornwall, Pa., which contained a percentage of copper that yielded about 0.5 per cent. in the steel; the analyses of the rails showed about 0.5 per cent. of copper, so Mr. McKenna thought that they would see what putting it in the steel in the above-described way would accomplish. Of course, what it will accomplish, if anything, remains to be seen.

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ELWOOD HAYNES, Kokomo, Ind.:—The inspection of the rails, as I understand it, applies to the finished metal, ready for making into rails, and then to the rail itself and to the method of treatment. It seems to me that back of that the process of making the steel is fully as important as the inspection of the ingot and the rail after it is worked out. The introduction of titanium or other purifying agents into the steel, it seems to me, would have fully as much effect as the proper treatment of the steel in rolling it into rails. In the automobile business I know that we are very particular to use, in certain cases where the material is subjected to vibration, either a nickel-steel or a vanadium-steel. Of course those materials would be too expensive for steel rails. I have made a great many personal tests of material under shock and under vibration. When perfectly good steel is placed under vibration a breaking-up of the fibers takes place after a little while, usually after 40,000 or 50,000 vibrations of a certain amplitude, as is well known, of course, to engineers, and the apparently good piece of steel breaks. The physical quality or its general appearance apparently makes little difference. If it contains a certain amount of vanadium or nickel, the break does not occur until after millions of vibrations. It seems to me that the inspection should go back to the composition of the rail, which is fully as important as the treatment.

MR. HIBBARD:—When shearing off the usual amount discloses unsound metal, and the shearing is continued until sound metal is reached, who determines when this additional shearing shall take place and what it shall amount to?

MR. HUNT:—That is a point which I am glad you raised. It has led to perhaps as much dispute in regard to the working of this inspection as any one point. It is a very delicate situation, as to how the inspector shall act under those conditions. I am insisting that it is not the inspector's duty; or, rather, under the existing contracts, the manufacturer would have a right to object. If the inspector should say to the shearsman, "You must continue cutting that bloom off, you have not cut it enough," etc., the responsibility then would be put on the inspector, and the manufacturer relieved from it, and his work-

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men relieved—as the shearsman cut as long as the inspector told him to, etc. The inspector's relations are not quite that, and I do not think it would bring as good results as if he simply satisfied himself that a certain ingot of Heat No. so-and-so had been improperly sheared, and immediately reported that knowledge to the foreman in charge of that department in somewhat this form: "Your shearsman did not properly shear that bloom or those blooms." Such a reported condition follows that bloom into the rail; that rail has that individual history attached to it, and it follows it into service, and the knowledge of such notation and report will, in my judgment, result in the workmen being more careful than if they could say, "Well, the inspector decided the bloom was sheared enough," or, "I cut it as far as the inspector wanted," etc. Proper shearing is often a difficult point to determine. As every steel-maker knows, the movement of the shear, its physical action on the steel, is very much like cutting a piece of cheese with a knife; it will carry the metal down and perhaps conceal a porous condition. A very pronounced case of unsoundness can be readily detected, but that condition is not always so evident.

R. H. Sweetser, Columbus, Ohio:—What is the limit of phosphorus in Bessemer rails, and is the upper limit more closely adhered to than it has been for the last several years in the United States?

Mr. Hunt:—Not to exceed 0.10 per cent. is the universal Bessemer specification. It seldom gets above 0.096 per cent.; in fact, I think the general average has been during the last few years perhaps a little lower than it was preceding that time. As a rough reason for it, without having data to back the opinion, I believe that the large consumption—or the large percentage of diversion, if I may so put it, of rail orders to the open-hearth process has relieved the strain on the Bessemer ores, so that those mills which were running on Bessemer rails have had less trouble in obtaining ores giving them phosphorus well within the 0.10 per cent. limit than was the case a few years ago when all the rails were made by the Bessemer process. At all events, I know the current reports are showing an average of somewhat lower phosphorus.

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Notes on Titanium and on the Cleansing Effect of Titanium on Cast-Iron.

BY BRADLEY STOUGHTON, NEW YORK, N. Y.

(Cleveland Meeting, October, 1912.)

[SECRETARY'S NOTE.—To avoid repetition of foot-notes, references to authorities are made in this paper by means of figures, referring to a numbered list in the appendix.—J. S.]

INTRODUCTION.

THIS paper gives the results: first, of a research into the somewhat extensive literature of titanium; and second, of a series of original tests upon the effect of titanium on iron castings. New investigations have also been made upon the effect of titanium on steel, but I hope to treat this subject in an independent paper at some later date, and, therefore, will not discuss the experiments in detail here.

The combined results of this study, which extended over a period of more than two years, can be briefly summarized as follows:

1. Steel or cast-iron, in which the titanium is properly proportioned, and which has been properly treated, is improved in strength, toughness, and durability against wear, such as, for example, the wear of railroad-rails, of steel and chilled-iron rolls, car-wheels, etc.

2. These improvements seem to be caused, not so much by the direct effect of titanium on the metal, as by its cleansing influence in removing harmful impurities, such as oxygen and oxides, nitrogen, occluded slags, and perhaps also sulphur. It appears also to reduce segregation, which would contribute to the same end.

3. In order that its effects may be fully realized, the treatment of steel or iron with titanium must be correctly performed. There are a few simple, but essential, details to be observed; if they are neglected, the best results cannot be expected.

To illustrate by an example: A series of tests on cast-iron

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was carried on in a German foundry, in which scientific apparatus and instruments were employed, and great care apparently exercised in executing the work, but the titanium treatment was performed without a knowledge of the conditions which should have prevailed, and, as a consequence, the results were inconclusive. It is as if one should attempt to boil water with the best adapted appliances obtainable; with scientific instruments to measure the temperature and regulate the operations, and with every condition fulfilled except heating the water to a high enough temperature. Obviously, the result would be quite as unsatisfactory at 211° F., as far as boiling water is concerned, as at 60°, although the former point is within a hair's breadth of success. So, in adding titanium to iron or steel, if it is put into the slag, instead of the metal, or if opportunity is not given for it to be absorbed and do its work, or if the steel is subsequently allowed to become oxidized, or if the treated metal is cast much too hot or much too cold—if any, or all, of these important details are neglected, we must not expect to get the best results.

History of Titanium.

Although the existence of the element which we now know as titanium was discovered and reported by McGregor[4A] in 1791, its real scientific record did not begin until 1794 and 1795, when a mineral specimen of rutile and two specimens of cyano-nitride of titanium were studied by Professor Klaproth,[5] and by Vauquelip,[7] Hecht,[7] and Haüy.[8] Professor Klaproth—the same who gave the metal uranium its name—called the new element "Titanium—in honor of the Titans, the first children of the earth."[6—p. 9] The first attempt to isolate it was made by Lampadius,[9, 11] who, in 1797, attacked a synthetic oxide by charcoal and heat, and obtained crystals of a dark copper color which he believed to be the metal, but which we know from repetitions of the experiment, and from the color of the product, to have been the cyanide or nitride, or both. In 1825, Berzelius[15] reduced the fluo-titanate of potassium—K_2TiF_6—by means of metallic potassium and heat, and obtained an amorphous gray powder, which is stated by Woehler[16—p. 181] to be the first specimen of metallic titanium produced. Subsequently, Woehler[16] himself, and other investigators who preceded him,

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obtained the metal by reduction with potassium, sodium, electrolysis, and other means; in each case the product was a black or grayish-black powder, or silver-white grains.[23] Notwithstanding these researches, the cyano-nitride of titanium, with its characteristic red color—said to be the only substance in nature resembling copper in appearance—was commonly believed to be metallic titanium until the publication of Woehler's classic monograph [16] in 1849. This paper of Woehler's contained the first published analysis of the copper-colored mineral, which had been originally studied in 1795, as mentioned above, and which had long been familiar to many iron-masters through its presence as accretions in blast-furnaces and as occluded crystals in slags.

So important had titanium become among the metals in 1864 that Rivot [1] devoted a chapter of his *Docimasie* to it, and, in 1879, Hallock [2] published a bibliography on titanium comprising several hundred references. In 1886 Shimer [33] further advanced our knowledge by showing that the traces of titanium rarely found in pig-iron occurred in the form of the carbide—TiC—at least in the five or six samples which he examined, and T. W. Hogg [37] extended this in 1893 by his investigations of ferro-manganese, in which he found millions of crystals of titanium cyano-nitride, with occasionally the nitride and carbide.

Rossi [3, 63, 110] has repeatedly shown that titaniferous iron-ores can be smelted successfully in the blast-furnace, but the presence of titanium compounds has nevertheless always been considered by most smelters a serious handicap. This phase of the subject was discussed by Riley [17] in 1863, Bowron [108 A] in 1882, and the first of Rossi's papers [3] appeared in 1890. This question forms a most interesting study by itself, but of late years has been somewhat superseded in the literature by accounts of the manufacture of ferro-titanium in the electric furnace. Before we come to that process, however, it is appropriate to mention the first production of melted specimens of metallic titanium, which was accomplished as a laboratory-experiment by Moissan [18] in his electric furnace in 1894.

Previous to 1895, Auguste J. Rossi [110] appears to have produced ferro-titanium, by an industrially practicable method. Its possibilities were, however, very slow in being recognized.

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What may be regarded as the first extended application to steel began at the Maryland Steel Works in November, 1907; to cast-iron,[91] at the car-wheel foundry of the Norfolk & Western railway at Roanoke, Va., in August, 1908. The rapidity of progress since that time may be learned from the statistics of the subsequent years' production: In 1911, in America alone 79 manufacturers of steel treated a total of 410,600 tons of steel with ferro-titanium alloy—the average being about 9 lb. of alloy (containing from 10 to 15 per cent. of titanium) per gross ton of steel. A large, but unrecorded, amount of cast-iron was also treated. The present importance of this treatment will be indicated by comparison with other steel-alloying elements; thus: more steel and more cast-iron was treated with titanium in 1911 than was treated with all the other elements combined, except manganese and silicon (aluminum).

Occurrence.

Titanium has often been spoken of as one of the "rare elements," but this description is true only in the sense that the metal is seldom met with in nature and in the arts under its own name and identity. This is because it unites with extraordinary eagerness with both the oxygen and the nitrogen of the atmosphere and with other elements and metals prevalent in nature, and because it can be separated from these combinations with much difficulty, so that it has never been found in the free state, and has only rarely been reduced from its compounds by artificial means. In every other aspect it is far from being rare: a reliable estimate places it among the ten commonest elements in the earth's crust, together with oxygen, aluminum, iron, silicon, etc.[45 A] It has often been recognized in plants, in the human blood, in meteorites, in the spectrum of the sun,[29] and elsewhere. It is frequently found in clays, where it has the effect of increasing their fusibility.[45] Numerous references tell of its usefulness in the arts for a hundred years back—long before electric smelting made it commercially available in metallurgy. Indeed, the widespread prevalence of titanium prevented giving herewith, as I had hoped to do, a comprehensive bibliography of the literature relating to it, because the publication in our *Transactions* of far more than a thousand references on the metallurgy and chemistry of tita-

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nium alone would scarcely be warranted. Even a list of references to the occurrence of titanium in connection with iron-ores would cover a period beginning in the eighteenth century,[10], [8] and would far exceed the space available here. Washington[46] reports that titanium is probably never wholly absent from any rocks, but that it is distinctly more abundant in basic than in acid rocks, and its affinities lie more with iron and sodium than with magnesium and, still less, with calcium; indeed, rocks high in calcium and phosphorus seldom contain much titanium. This has often been noted in iron-ores, which are more free from phosphorus when associated with titanium.

From the standpoint of the blast-furnace manager, iron-ores are not classed as "titaniferous" unless they contain more than 2 or 3 per cent. of titanium, because these small amounts give no especial difficulty in smelting; but vast deposits—even mountains—of iron-ore lie idle in various parts of the world[42, 49, 3 (pp. 93-4), 2] because they contain from 5 to 50 per cent. and more of titanium oxide or of the so-called "titanic iron"—a mineral whose formula is stated by Koenig and von der Pforten[36] to be $FeTiO_3$. Most of these titaniferous ores are magnetic, hematite being only infrequently accompanied by titanium.[43] The best known deposits are in Scandinavia,[40, 44, 45] Canada,[32] New York State,[39, 30] Lake Superior,[10] Australia,[27] and Wales.[41]

In the smelting of ores containing titanium, the latter usually passes more or less completely into the slag in oxidized form.[37] However, it does not require a higher temperature than is often employed in the iron blast-furnace to reduce oxide of titanium to the nitride, cyanide, or carbide,[9, 13, 14, 16] and accretions of the cyanide and nitride are often found in the hearth, as well as occasionally at higher levels, its presence in the latter location being due, no doubt, to the volatility of the cyano-nitride and its condensation in cooler regions.[66, 69 A, 79 (p. 482)] Microscopic crystals of cyanide and nitride are often seen in blast-furnace slags[22, 23, 31] and have been found in ferro-manganese, as already mentioned.[37] I am indebted to J. E. Johnson, Jr., for the following explanation (in a private communication of July 18, 1912) why cyano-nitride is more frequently found in ferro-manganese than in pig-iron:

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"Titanium is a highly refractory substance, and in small quantities is normally slagged out of the furnace. In order to make spiegel, and particularly ferro-manganese, it is necessary to run the furnace very hot and also very basic to prevent scouring of the manganese into the slag. I should say that these were the two conditions best calculated to reduce the rutile to cyano-nitride."

As regards the presence of metallic titanium in the elemental condition either in blast-furnace slag or in pig-iron, I have been unable to find any evidence to indicate that it occurs as such. Wollaston, in his oft-quoted treatise,[22] speaks of "metallic titanium found in blast-furnace slag," but his paper shows that he mistook the cyano-nitride for the metal, as most investigators did previous to 1849. De Greswolde,[23] who followed Wollaston, and in 1829 dissolved a titaniferous blast-furnace slag in aqua regia, did indeed obtain black and silver-white metallic powder and grains, but Klaproth,[5, 6] had already shown in 1795 that both nitric and hydrochloric acids would dissolve the cyano-nitride, which may therefore have been the origin of any metallic titanium fortuitously produced by de Greswolde, if, indeed, his grains were titanium. The methods of analysis used on pig-iron would not distinguish between titanium in the elemental state, or as cyanide, nitride, or carbide, and the many occurrences of titanium in pig-iron reported from the first half of the nineteenth century onward, indicate nothing more than that titanium was present in some form or other. Even as early as 1841 Karsten[51—p. 534] expresses the opinion that the titanium present was probably mechanically mixed, and not combined with the iron. Wedding[61—Vol. I., p. 373] expresses a like opinion. Percy,[54] who quotes several analyses, says that it appears to be present as TiO_2, while Bell[58—pp. 157, 167] mentions the presence of "titanium or its nitride." In view of the irreducibility of compounds of titanium by carbon at temperatures higher than those of the blast-furnace, even in the presence of iron, I am inclined to agree with Rossi that the small amount of titanium found—only rarely exceeding 0.50 per cent. and somewhat doubtful when over 1 per cent.[81]—was always in the combined state.

Physical Properties of Titanium and Its Compounds.

Weiss and Kaiser[4—p. 392] determined the melting-point of a mass of impure titanium containing Ti, 95.1; Al, 2.78; H,

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1.07; C, 0.42; undetermined, 0.33 per cent., and report it somewhat vaguely as between 2,200° and 2,400° C. (3,992° and 4,352° F.). This figure has often been quoted as the melting-point of the element, but, obviously, the presence of the impurities mentioned would alter the fusing-point materially. Both Moissan and Rossi, who have probably had more experience with the metal in molten masses than any other scientific investigators, give much higher figures. Moissan says[18] that titanium with 2 per cent. of carbon is the most refractory substance he has met with in his electric-furnace experiments, and Rossi[110] affirms that the melting-point of titanium is between 2,700° and 3,000° C. (4,892° to 5,432° F.).

However, Hunter[116] contradicts both these estimates, and explains them on the ground that Moissan and others produced, not pure metal, but alloys of the metal with its compounds. His estimate of the melting-point is between 1,800° and 1,850° C.

Weiss and Kaiser give the following figures for constants of what they call their "pure metal"[4—p. 395]—*i. e.*, Ti, 97.41; H, 1.56; C, 0.30; undetermined, 0.73 per cent.; specific gravity, 5.174; specific heat, 0.1418; atomic heat, 6.830; heat of combination with oxygen: $Ti + 2O = TiO_2 + 97{,}772$ calories. I much favor, however, the figures for specific gravity given by Moissan, Rossi, and Hunter; the first-named gives 4.87;[18] the second, 4.85[110] and 4.84, and, finally, Hunter says 4.50.[116—pp. 335-6] The color of the metal in massive form is silver-white, with a brilliant white fracture.[4, 18] It is hard and brittle when cold, but can readily be forged at red heat.[116—pp. 335-6] The melting-point of ferro-titanium, made by the thermit process, and containing from 20 to 25 per cent. of titanium, is said to be between 1,300° and 1,400° C. (2,372° and 2,552° F.).[119 A]

Titanium seems to give extreme hardness to some of its alloys and compounds; thus, its ternary alloy with aluminum and copper is said to be very hard and resilient; the cyanides, nitrides, and carbides are very hard, indeed. The cyano-nitride, according to Vauquelin and Hecht,[7] is hard enough to scratch glass, but is pulverized with difficulty.[7—p. 11] According to Rivot,[1—p. 131] it will even scratch quartz, but, on the contrary, he says that it is very fragile, and is pulverized with ease in a mortar. Lampadius[11] found it brittle, but possessing in

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small scales a considerable degree of elasticity. These differences, as well as others to be noted, are no doubt explainable, in part at least, by the variable composition of the cyano-nitride, which fluctuates greatly in the relative proportions of cyanide and nitride contained; for example, the specimen examined by Woehler,[16] whose analysis has been so often quoted, contained Ti, 77.26; N, 18.30; C, 3.64 per cent., while the crystals separated by Hogg from ferro-manganese [37] contained from 60.5 to 79.8 per cent. of Ti, and always some iron. The cyano-nitride has a brilliant metallic luster, and a markedly characteristic red-brown, or copper, color. It crystallizes often in cubes, sometimes as small as 0.001 in. in diameter; less often in octahedra and other isometric forms; it is not easily fused,[11, 67] but volatilizes at the temperature of the crucible-steel melting-furnace [66, 69 A, 79] (which would be about 1,750° C.). Its specific gravity is given by different observers as follows:

1796, Vauquelin and Hecht, reference No. 7, specific gravity.....4.2469
1822, Wollaston, reference No. 22, specific gravity.....5.3
1897, Franck, reference No. 67, specific gravity.....5.28
1895, Hogg, reference No. 37, specific gravity.....4.1 to 5.1

Shimer gives the specific gravity of the carbide—TiC—as 5.1.[33] Moissan[18A] gives it as 4.3. Wollaston[22] says that the electric conductivity of the cyano-nitride is great.

Titanium has the power of forming compounds having a variety of colors of unusual richness, such as red, green, yellow, blue, etc. For instance, the different nitrides studied by Woehler were violet, golden yellow, and copper colored. In this property titanium resembles gold. The compounds of titanium also vary greatly in their state of aggregation—from the extraordinary infusibility of the cyano-nitride, to the fluidity of the chloride, and the volatility of the chloride [68] and hydrofluoride.

Chemical Properties of Titanium.

The chemical properties of titanium are, in general, similar to those of silicon, with the difference that titanium is more active in combination, both as to the number of elements with which it unites readily, and the number of different compounds formed with each. In some respects it is also far more energetic in its action than silicon; as, for example, it unites with

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oxygen, nitrogen, chlorine, and bromine with so much energy as to become incandescent. [16 (pp. 181-2), 18, 73 (p. 126)]

It unites readily with oxygen at 610° C. (1,130° F.) and this reaction, according to Kaiser,[4] evolves 97,772 calories. Had Kaiser experimented with purer material this result might have been different. It is one of the elements like sodium, etc., which, when brought into contact with water, decompose it to form hydrogen gas and metallic oxide.[77 A, 16 (p. 182), 59 (p. 164)] Upon warming powdered titanium, it unites readily with oxygen with brilliant scintillation, and the reaction is so energetic that it propagates itself thereafter, even if removed from contact with the flame.[16—p. 181] Woehler states that he does not know of another body which burns as energetically in oxygen as titanium, and furthermore, that metallic titanium mixed with oxide of copper and warmed, reduces the copper to the metallic state with the evolution of great heat.[16—p. 182] It forms many oxides, such as TiO, Ti_2O_3, TiO_2, etc., of which TiO_2 is the most common.[1 (p. 122), 79, et al.] It also forms oxy-chlorides.[70] The oxide of titanium is reduced by aluminum with evolution of heat.

Titanium has a greater affinity for nitrogen than has any other known element. When heated to 800° C. (1,472°F.) in an atmosphere of nitrogen gas, it burns brilliantly with incandescence. Roberts-Austen [73] states that this is the only known instance of vivid combustion in nitrogen. It forms several nitrides, such as Ti_3N, TiN, Ti_2N_2, Ti_3N_4, Ti_3N_5, etc.[16 (p. 180), 70, 116] The massive and fused titanium is not so active in uniting with nitrogen as the powder.[18] Titanium oxide, when heated in the presence of nitrogen, decomposes into titanium nitride; [79 (p. 480), 79 A, 18, 116, 4]—and titanium cyanide [1 (p. 131), 16, 71, 79 B] if carbon also is present. It is in this way that the various mixtures of nitride and cyanide, somewhat vaguely termed "cyano-nitride of titanium," are produced in the blast-furnace and elsewhere. Even titanium cyanide will absorb nitrogen with the evolution of heat.

Titanium also forms a stable carbide, TiC, which has been found in pig-iron and ferro-manganese, but which is said to be almost absent from ferro-titanium, that is to say, from pig-iron when the titanium is large in amount—say more than 10 per

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cent.[81A] Indeed, the action of titanium is to precipitate carbon from cast-iron and steel.[81A]

Titanium forms one stable sulphide :—TiS_2[1—p. 132]. Blair and Shimer found crystals of the following composition in a gray pig-iron : [81B] Ti, 62.82; S, 22.64; C, 9.82; Fe, 1.82; undetermined, 2.90 per cent. I am inclined to think that titanium sulphide is not readily soluble in molten iron and steel, but, like manganese sulphide, it separates from the molten bath. But the action seems to be more rapid in the case of titanium sulphide than in the case of manganese sulphide, because Goldschmidt [122] and Treuheit [126] found the amount of sulphur decreased after a brief titanium treatment.

Titanium unites with chlorine, fluorine, iodine, hydrogen, etc.; also it forms a silicide, aluminide, boride, etc., but has only slight affinity for phosphorus and arsenic.[1 et al.]

The oxide of titanium acts as an acid flux and unites with iron oxide, aluminum, calcium and other metals to form titanates.[81C] These titanates apparently lower the melting-point of silicates of the same metals; *i. e.*, the dirt and slags occluded in iron and steel.[81C]

Titanium in Pig-Iron.

Some writers have placed a good deal of emphasis on the circumstance that certain grades of pig-iron which have borne an unusually good reputation for strength and toughness for a long period of years, have been thought to contain traces of titanium, or else have been smelted from mixtures containing titaniferous ores. Rossi [8—pp. 102-3] gives a list of references relating to this subject, and a number of text-books freely quote such statements.

It is not to be denied that a reputation for excellence extending over a long period of years forms some basis for such an opinion, as, for example, the titaniferous charcoal pig-iron of Turrach, which has borne an excellent reputation for 250 years.[64] Neither is it to be denied that some of the most careful metallurgists of the nineteenth century have attributed the excellence of certain grades of pig-iron to titanium, or have, at least, attributed excellence to titaniferous pig-iron; for example: Bell,[52—pp. 750-1] Rivot,[1—pp. 156-7] Osborn,[56—p. 475] Crookes and Roehrig,[55] Åkerman,[57] Bauerman,[59—p. 53] Raymond,[63A] and

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others. Finally, it is admitted that the presence of titanium in the blast-furnace, or the presence of compounds of titanium in pig-iron, might have the effect of opposing the entrance of nitrogen to the metal, which would, doubtless, increase its strength and decrease its brittleness. Nevertheless, all this evidence is necessarily circumstantial, and in case of a conflict of testimony, I should be inclined to place more weight upon comparative tests of cast-iron, such as those given in a later section of this paper.

Titanium in Steel.

What is now often called "titanium-steel," is not a true alloy-steel in the same sense as nickel-steel, manganese-steel, etc., because the titanium is added for the purpose of cleansing and purifying the metal, and not with the expectation of producing new properties by its direct influence. On the contrary, when sufficient titanium has been added to produce a direct influence, the result has not always seemed favorable.[86] Indeed, we may make excellent titanium-steel in which not a trace of titanium can be found—it already having performed its scavenging work and passed out of the bath together with the impurities it was commissioned to remove. The case is somewhat similar to that of manganese, which is added to almost all steels for the purpose of removing oxygen and sulphur, and neutralizing the effects of any traces of those elements which remain thereafter. Small doses of manganese are used for this purpose. However, when we wish to take advantage of the direct effect of manganese, we add about 15 per cent. of the metal, and then get an entirely new result, which makes manganese-alloy steel. Most of the titanium-steel belongs in the first class, and not in the true alloy-steel class. Indications are not wanting, however, that the direct effect of titanium on steel and iron may have its advantages,[100-104 A] and the future will perhaps see titanium alloy-steel used for many purposes.

Even when used for cleansing purposes, titanium hardens, strengthens, and toughens steel, and this seems to apply to crucible-,[91 A] and even electric steel,[91 A] as well as to Bessemer [94,97,99] and open-hearth.[98,103] Touceda tested two steels in a Wright-Souther alternate testing-machine: one of them had been treated

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with 0.25 per cent. of the usual 10 per cent. grade of ferro-titanium alloy, and the other was like it but without having been so treated. The untreated steel stood 2,676,000 revolutions at a maximum fiber-stress of 38,872 lb.; the treated steel stood 18,274,000 revolutions at a fiber-stress of from 38,872 to 45,939 lb. Waterhouse tested the segregation of steel which had been treated with titanium, and that which had not been so treated, and found a reduction of segregation in the former.[102]

Titanium and Non-Ferrous Metals.

Titanium hardens and strengthens aluminum and increases its toughness and resiliency. It is stated that the addition of titanium, chromium, and copper to aluminum gives some of the hardest and toughest light alloys yet produced. Some years ago, small percentages of titanium were added to aluminum to make an alloy which was used by European bicycle manufacturers; the strength of the aluminum was increased from 23,300 to 73,500 lb. per sq. in.[105] Richards states that titanium has been alloyed with aluminum in proportions up to 7 per cent., but that 2 per cent. gave the best result. This alloy had an elasticity comparable to spring-brass, but it corroded badly, and so has disappeared from the industrial world.[107] Copper is greatly improved by the addition of titanium in small amounts to remove oxygen and gases, and this is a branch of the industry which is now undergoing rapid development.

The Manufacture of Titanium and of Ferro-Titanium.

The manufacture of titanium in a substantially pure state by a process cheap enough to be commercially satisfactory, has apparently not yet been accomplished. Moissan's method of reducing the oxide by carbon in an electric furnace seems industrially impracticable; the alumino-thermic process of Goldschmidt incorporates aluminum as an impurity; finally, any smelting process not carried on *in vacuo* gives the molten metal an opportunity to absorb material amounts of nitrogen and oxygen. Numerous wet processes have been developed in laboratories, and have produced samples of titanium of varying degrees of purity from 95 per cent. upward, but these are all too costly to be adopted commercially unless the use to which the metal is to be put will justify a high price. Weiss and

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Kaiser[4] and Hunter[116] have described these several methods in some detail, and the latter has obtained a product in which he found by two different methods of analysis nearly 100 per cent. of Ti. His process, in brief, is the reduction of titanium tetra-chloride, $TiCl_4$, by sodium at a red heat. Before his experiments were made, Georges and Staehler[114] had reduced the tetra-chloride by means of hydrogen, but sodium would of course give a purer product, although a more costly one. Indeed, the reduction by means of sodium is not new with Hunter,[2] but it, like all these laboratory methods, is far too costly to produce a metal that can be used in the manufacture of iron and steel. For this purpose we must employ some process comparable in cost with the Goldschmidt thermit process, or the ferro-titanium process of Rossi, which two methods of manufacture are the only ones which have been used on a large scale in connection with metallurgy, so far as I know, although Huppertz[20] and Borchers,[118] Lodyguine,[111] and Gin[112] have each developed processes which seem to have important possibilities.

The Goldschmidt thermit process, as applied to titanium reduction, is not a simple application of the base reaction, as follows:

$$3 : TiO_2 + 2 : Al_2 = 2 : Al_2O_3 + 3 : Ti,$$

because this would not evolve sufficient heat; one employs instead a mixture of iron thermit with titanium thermit in such proportions that, when ignited, a ferro-titanium will be produced containing Fe, 75; Ti, 25 per cent. thus:

$$4 : Fe_2O_3 + 4 : Al_2 = 4 : Al_2O_3 + 4 : Fe_2;$$
$$3 : TiO_2 + 2 : Al_2 = 2 : Al_2O_3 + 3 : Ti.$$

The heat evolved by the first reaction augments that evolved by the second reaction, and this makes an energetic exothermic effect. Ferro-titanium could, if desired, be made in this manner independently and then added to iron or steel, but it is not applied this way industrially. Instead, one step is made of the two operations, and the mixture of iron- and titanium-thermits is added to the molten iron or steel which is to be treated, so that the alloy will be reduced *in situ*.

Rossi makes ferro-titanium containing from 10 to 15 per

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cent. of titanium, and from 5 to 8 per cent. of carbon, and the residue iron, by heating a pulverized mixture of titaniferous iron-ore and charcoal to a temperature of not less than 1,927° C. (3,500° F.). This operation is performed in an arc electric furnace of the Siemens type. If an alloy low in carbon is desired, a slightly different form of furnace is employed, in which graphite does not come in contact with the molten alloy, and the reduction is performed by liquid aluminum, as follows: Scrap-aluminum is charged and melted, followed by the desired amount of scrap-iron. The iron forms a liquid bath upon which the aluminum floats. Upon this layer of aluminum is charged rich titaniferous ores, or else a titaniferous slag consisting of calcium titanate, the manufacture of which is performed in an auxiliary furnace. The reaction between the melted aluminum and the titanium oxide occurs at once at the surface of contact, and has so much energy of its own that little electricity is required thereafter to keep the temperature up to the reaction-point. The reduced titanium unites with the iron bath. The rich slag used in this operation is also used when desired instead of titaniferous iron-ores in making the high-carbon grade of ferro-titanium. It is made by smelting titaniferous iron-ores with a limited amount of charcoal and at a temperature sufficiently elevated to reduce iron and silicon, but not to reduce titanium. Pig-iron substantially free from titanium is thus produced, and contains the bulk of the impurities in the ores, such as sulphur, phosphorus, etc. The titanium oxide is fluxed with lime to form a slag consisting chiefly of calcium titanate, CaO, TiO_2.

Researches Upon the Effect of Titanium on Cast-Iron.

In some of his experiments, Rossi appears to have added 5 per cent. of a pig-iron containing 0.2 per cent. of titanium to a non-titaniferous pig-iron, and increased the transverse strength thereof from 3,390 to 3,778 lb. per sq. in. Then he added 25 per cent. of the same pig-iron, but the strength was increased only to 3,480 lb. I am inclined to believe that he chilled his iron when using the larger amount of titaniferous pig, or else that he raised the freezing-temperature without equally raising the casting-temperature, and thus destroyed in part the good effect on strength which the titanium treatment would have had.

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This matter of the casting-temperature of cast-iron is one which a great many investigators of titanium treatment seem to have overlooked; indeed, it is one upon which little scientific thought appears to have been expended, for I have found few published data upon it. However, Longmuir [121] has studied the effect of casting-temperature on tensile strength, and his figures will serve as a guide to show us the importance of this question. I have averaged the record of about 42 tests published by him in 1903 and 1904, with the following result: The strength of iron cast at the correct temperature (estimated by Longmuir as about 1,230° C. = 2,246° F.) averaged 23,985 lb. per sq. in.; that of iron cast too hot (estimated by him as about 1,320° C. = 2,408° F.), was 16,100 lb., equivalent to a loss of 33 per cent.; that of iron cast too cold (estimated as about 1,120° C. = 2,048° F.), was 17,781 lb., or a loss of about 25 per cent. Therefore, if we are aiming to improve strength, we must guard against casting the iron either too hot or too cold, and this matter is equally important whether we operate by means of titanium or in any other way.

Rossi's experiments above noted appear to have been made in crucibles and in the iron casting-ladle. He supplemented them by others in which he used 4 per cent. of a ferro-titanium alloy containing from 10 to 15 per cent. of Ti. By this means he increased the transverse strength of cast-iron by from 20 to 25 per cent., and the tensile strength by from 30 to 50 per cent.

These tests were followed by those of Schiemann,[120] who added 5 per cent. of ferro-titanium thermit to a series of cast-irons, and increased the tensile strength by varying proportions between 30 and 50 per cent.

Rossi made some experiments in 1905, in which he added 1 per cent. of a ferro-titanium alloy containing 12 per cent. of titanium to cast-iron and increased the transverse strength by from 17 to 23 per cent., and the tensile strength by from 6 to 29⅛ per cent.

In the same year, Goldschmidt [122] reports a series of tests in which he added 0.1 per cent. of titanium to iron by means of thermit, and obtained increases in the transverse strength of from 1 to 17 per cent., and in the tensile strength of from 4 to 16 per cent.

In 1908, Prof. Martin Hokanson, [123—pp. 62-3] of the Carnegie

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Technical Schools, found the compressive strength of chilled cast-iron which had been treated with ferro-titanium alloy to be 298,000 lb. per sq. in., as compared with 173,000 lb. for the same metal untreated. The hardness of the treated metal was 557 (Brinell), as compared with 455 for the untreated iron.

Dr. Richard Moldenke [123] made a series of transverse tests on white and gray cast-irons in 1908, with the following results:

Gray Irons. Lb. Per Sq. In.	White Irons. Lb. Per Sq. In.
Untreated iron (average of 9 tests), 2,020	(Average of 8 tests)..... 2,050
Plus 0.05 p.c. of Ti[a] (average of 4 tests), 3,100	(Average of 11 tests)..... 2,400
Plus 0.10 p.c. of Ti (average of 3 tests), 3,030	(Average of 9 tests)..... 2,420
Plus 0.05 p.c. of Ti (average of 6 tests), 3,070	(Average of 10 tests)..... 2,400
Plus 0.10 p.c. of Ti (average of 6 tests), 2,990	(Average of 10 tests)..... 2,520
Plus 0.15 p.c. of Ti (average of 4 tests), 3,190	

[a] Titanium added in form of ferro-titanium alloy.

Average of treated gray cast-iron, 3,070, equivalent to increase of 52 per cent.

Average of treated white cast-iron, 2,430, equivalent to increase of 18 per cent.

Moldenke added ferro-titanium alloy equivalent to 0.10 per cent. of titanium to the bed of a cupola, but the results of tests of the metal therefrom were so irregular that he judged an average to be unrepresentative. I have duplicated this result several times, and am inclined to believe that the oxidizing influence brought to bear on the bed of a cupola makes the use of titanium there of uncertain value as far as strength is concerned. The whole question of the use of titanium in the cupola requires investigation. The ferro-titanium alloy melts at a higher temperature than the iron itself, and therefore gets nearer the tuyeres before it begins to trickle down into the hearth. If, during this transit, it meets with slag, its value will be lost as far as the iron is concerned, because it will attack the oxygen in the slag and unite with it, thus being used up. Cupolas must be constantly drained clear of slag when titanium-alloy is melted with cast-iron therein. We must go even further than this precaution to prevent a layer of slag accumulating on top of the metal and intercepting the drops of melted alloy, because constantly draining the cupola will not keep the slag away provided the level of the iron bath rises and falls several inches at intervals, as, for example, when the iron is collected inside the cupola instead of in the ladle.

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Feise [124] gives the following table of tests made on gray cast-irons:

	Transverse Strength.	Deflection.
Without titanium	1,230 kg/qcm.	0.046 mkg.
Plus 0.25 per cent. of Ti	1,660 kg/qcm.[a]	0.056 mkg.
Plus 0.50 per cent. of Ti	1,410 kg/qcm.	0.053 mkg.
Plus 1.00 per cent. of Ti	1,340 kg/qcm.	0.066 mkg.

[a] Equivalent to 35 per cent. increase in strength.

These results corroborate the observations of others: that the maximum effect on strength appears to be obtained with about 0.10 per cent. of titanium.

The most unsatisfactory results I have found were published by Treuheit,[126] and report an elaborate series of tests on cast-iron treated with ferro-titanium thermit and ferro-titanium alloy, respectively. The temperatures were measured by a pyrometer, and the resulting metal was tested by both tensile and transverse methods. In some cases the treated metal was weaker than that which had not been treated, and sometimes a little stronger, but never enough different to indicate anything especially interesting. In examining the record of this work, one does not have to look far to find a reason for the unsatisfactory result. The temperatures at which the iron was cast were widely variant, sometimes being above 1,300° C. (2,372° F.), which is shown by the work of Longmuir to be detrimental to quality. One might also infer from the author's words that he did not allow the metal to stand long enough after adding the titanium, so that the latter could exert its cleansing influence; at best, his record is not clear on this point. Three minutes should be allowed the titanium to act upon cast-iron before it is teemed, and more if the alloy is not then absorbed, as indicated by the pasty state of what remains. If the iron has not heat enough to permit this 3-min. wait, then it should have been produced in hotter condition in the first instance; but, in any event, if it cannot be held 3 min., and still poured at the correct teeming-temperature, one must not blame the lack of success on the treatment, but on the manner in which the treatment is performed. I have failed in test after test because the metal had to be carried some distance from the cupola before it could be treated, or because it was at about the correct pouring-temperature before the 3-min. wait, and, there-

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fore, too cold thereafter. If the titanium treatment is advantageous to cast-iron, it must also be advantageous to perform the treatment properly.

When titanium unites with the oxygen or nitrogen in iron or steel, there is, unquestionably, an evolution of heat, which opposes the cooling of the bath during the wait. Fitzgerald [123] showed that the temperature of a bath of cast-iron increased 25° F. during its treatment with 1 per cent. of titanium in the form of ferro-titanium, but that this lasted only 1 min., and then the temperature slowly fell again. The original temperature of the metal must be hot enough to endure this drop and still be at the correct point.

Another mistake made by Trueheit, when treating this iron with ferro-titanium alloy, was in heating the alloy red-hot in the ladle before tapping the iron therein. This heating has the effect of oxidizing the alloy and reducing, or, maybe, nullifying, its cleansing effect; the practice was once recommended by the manufacturers of the alloy, but is now abandoned. Furthermore, the alloy should not be placed in the bottom of a ladle, because its infusibility causes it to become pasty and stick to the lining without dissolving in the bath. It should be thrown into the ladle after there is a layer of iron upon which it can float, and then the in-pouring stream of metal will stir it well and assist its absorbtion.

If Trueheit's experiments were the only ones to guide us, or if they were confirmed by other results skillfully obtained, then we might doubt the value of titanium to cast-iron, but the many favorable researches quoted herewith would make us question Trueheit's conclusions, even if we could not point out very definitely the cause of his ill-success.

C. H. Gale,[127] one of the most experienced American malleable cast-iron manufacturers, reported, in 1911, the results of many tests made on the treatment of malleable cast-iron with ferro-titanium alloy. In his first test, Gale added, first 1.25 and then 2.50 per cent. of the alloy in the ladle, with the consequence that the iron was teemed much too cold, and, also, it proved to be mottled, instead of having the usual, and desired, white fracture. Both of these conditions would, of course, decrease the strength of the metal; nevertheless, the treated bars in one set averaged slightly stronger than those untreated.

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In another set they were about 19 per cent. weaker in tensile test. The transverse tests of the treated metal proved it to be from 5 to 30 per cent. stronger than the untreated metal. Nevertheless, the result indicated that such a ladle treatment of malleable cast-iron did not promise much success unless the silicon were reduced so as to prevent mottled-iron being produced, since titanium causes a precipitation of graphite, and unless the metal were increased in temperature so as to wait during the time of treatment and still be at a good heat for pouring. (I may say here that a test which I made on malleable cast-iron fully corroborates this conclusion.)

Gale then made a series of tests, adding the alloy in the furnace from 30 to 45 min. before tapping, but it scarcely seems likely that this would have as marked a result as if the alloy had been put in the furnace as soon before tapping as practicable, because titanium oxidizes so quickly that its effect would be lost in a comparatively short time. However, the transverse tests show an improvement on the average of 15 per cent. in the bars treated with 0.30 per cent. of the alloy, and 17 per cent. in the bars treated with 0.60 per cent. The tensile tests were not so favorable—12 per cent. and ⅛ per cent. respectively—but Gale cautions us against taking the tensile tests too seriously, because the bars were round in section and may not have been sound. In the discussion of those results, W. D. Alexander gave the report of a long series of tests made by him upon the treatment of malleable cast-iron with the usual ferro-titanium alloy—*i. e.*, that containing from 10 to 15 per cent. of titanium—in proportions from 0.47 to 1.75 per cent. He added the alloy in the hand-ladle caught from the furnace, and thus there was a great cooling of the metal, especially when using the larger amounts of alloy. The ductility of the metal was then tested, after annealing, by upsetting bars with a 10-lb. sledge. The metal treated with 0.47 per cent. of alloy showed an increase in ductility by this test of 105.3 per cent.; that treated with 0.75 per cent. of alloy improved 65.7 per cent.; that with 1 per cent. of alloy decreased 10.5 per cent.; that with 1.25 per cent. of alloy decreased 69.1 per cent.; that with 1.50 per cent. of alloy decreased 59.4 per cent.; and that with 17.5 per cent. of alloy decreased 128.2 per cent. Evidently the best result would be obtained with 0.47 per cent. of alloy or less, unless

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some means could be had of heating the metal hotter before the treatment.

In 1911, Geiger [128] published his book on iron- and steel-foundry practice, in which he devotes two sections to the effect of titanium on the ferrous metals. Finally, he reports a series of tests on iron with titanium in proportions of 0.04, 0.07, and 0.10 per cent., equivalent to 0.40, 0.70, and 1.00 per cent. of the usual ferro-titanium alloy, respectively. The respective improvements in tensile strength were 12, 24, and 26 per cent.; of transverse strength, 6, 9, and 10 per cent.; and of deflection in transverse test, 9, 11, and 6 per cent.

Thomas D. West,[129] by adding 12 oz. of the usual ferro-titanium alloy to a ladle of cast-iron containing 225 lb., obtained an increase of 27 per cent. in the transverse strength, as an average of five tests; by adding 22 oz. of ferro-titanium alloy to 225 lb. of iron, he obtained an increase in transverse strength of 32 per cent., as an average of five tests.

Osann [130—p. 118] reports on the improvement in the compressive strength of cast-iron for car-wheels; in one case an increase of 18 per cent. was obtained by treatment with titanium, and, in another case, of 75 per cent.

In my own tests, which included both white and gray cast-iron, the iron was melted in a cupola (except for one test on malleable cast-iron, referred to just above), and cast into round bars 18 in. long and from 1 to 1.3 in. in diameter. They were tested transversely on supports 12 in. apart, according to the standard testing-specifications of the American Society for Testing Materials and the American Foundrymen's Association. The figures reported below indicate moduli of rupture in pounds per square inch. During the research, I made several errors in operation, due to ignorance of the conditions which should have prevailed, but the reports of these improperly-performed tests are here given with the others, because, if they teach nothing useful, they at least emphasize the need of performing the titanium treatment correctly. The treatment was made with the usual ferro-titanium alloy containing from 10 to 15 per cent. of titanium, and the additions are recorded in percentages of alloy added, instead of percentages of titanium.

Considering first some of the incorrectly-performed tests, I made two tests charging the alloy with the iron of the bed of

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the cupola, hoping that the deoxidizing effect of the titanium would overcome the oxidizing effect of the blast. This hope was not realized, as will be seen below:

Test No. 1 (*Cupola Bed*).

Iron without titanium (average of 4 tests)..... 42,175 lb.
Iron with 0.50 p. c. of alloy on bed (average of 4 tests), 40,493 lb., decrease of 4 p.c.

This failure of the titanium-treated iron to exceed in strength that not treated is the more significant because the same iron showed an increase of 6 per cent. in strength when 0.5 per cent. of alloy was added to the ladle, although it had to be carried some distance in the foundry before pouring, and was therefore too cold after the treatment. The untreated iron was not bed-iron, and this difference is important.

Test No. 2 (*Cupola Bed*).

Iron without titanium (average of 4 tests).....41,520 lb.
Iron with 0.40 p. c. of alloy on bed (average of 4 tests), 40,610 lb., decrease of 2 p.c.

In spite of the apparent decrease in the treated metal, the titanium is shown to have been beneficial, because, normally, bed-iron would have been from 20 to 30 per cent. weaker than the other. This same iron gave an increase in strength of 16 per cent. when treated with 0.5 per cent. of alloy in the ladle, and of 9 per cent. when treated with 2 per cent. of alloy in the ladle.

I tried the experiment of adding the alloy in a cupola from which the slag was not drained continuously. The results were irregular and inconclusive, as shown below:

Tests Nos. 3, 4, 5, *and* 6 (*Wasted in Cupola Slag*).

Iron without titanium (average, 4 tests).....42,378 lb.
Iron with 0.5 p. c. of alloy (average, 4 tests)....35,775 lb., decrease of 15 p.c.

Iron without titanium (average, 6 tests).....48,517 lb.
Iron with 0.5 p. c. of alloy (average, 6 tests).....54,830 lb., increase of 13 p.c.

Iron without titanium (average, 8 tests).....44,893 lb.
Iron with 0.5 p. c. of alloy (average, 8 tests).....42,856 lb., decrease of 5 p.c.

Iron without titanium (average, 2 tests).....55,280 lb.
Iron with 0.5 p. c. of alloy (average, 2 tests).....52,570 lb., decrease of 5 p.c.

Several tests were made on iron which had to be carried

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some distance while molten, and which before treatment was at a good temperature for teeming, but cooled too far during the 3-min. wait so that it was too cold after treatment. At first the treatment was performed in a ladle holding about 225 lb. in which the loss of heat by radiation was very large. The results were:

Test No. 7 (*Iron Too Cold*).

Iron containing: Si, 2; S, 0.075; P, 0.73; Mn, 0.57 per cent.
Iron without titanium (average 2 tests).....55,280 lb.
Iron with 0.5 p. c. of alloy in ladle (average 2 tests)...50,100, decrease 10 p.c.

Test No. 8 (*Iron Too Cold*).

Iron containing: Si, 1.25; S, 0.040; P, 0.20; Mn, 0.70 per cent.
Iron without titanium (average, 3 tests).....49,730 lb.
Iron with 0.25 p.c. of alloy in ladle (average, 4 tests), 44,680 lb., decrease 10 p.c.
Iron with 0.5 p. c. of alloy in ladle (average, 2 tests), 46,000 lb., decrease 8 p.c.
Iron with 1 p. c. of alloy in ladle (1 test only).....54,300 lb., increase 9 p.c.
Iron with 2 p. c. of alloy in ladle (1 test only).....47,850 lb., decrease 4 p.c.

If the metal had been poured as cold as this, and had not been treated, the decrease in strength would have been fully 25 or 30 per cent. (see Longmuir, *ante*).

After this the iron was treated in a 1-ton ladle, in order to reduce radiation of heat, but still the metal was cool after the 3-min. wait, and the results were only partly satisfactory.

Test No. 9 (*Iron Cool*).

Iron containing Si, 1.50; P, 0.75 per cent.
Iron without titanium (average, 6 tests).....48,517 lb.
Iron with 0.5 p.c. of alloy in ladle (average, 5 tests) 49,280 lb., increase 2 p.c.

Test No. 10 (*Iron Cool*).

Iron containing Si, 1.30; P, 0.37 per cent.
Iron without titanium (average, 8 tests).....44,893 lb.
Iron with 0 5 p. c. of alloy in ladle (average, 7 tests), 48,419 lb., increase 8 p.c.

Test No. 11 (*Iron Cool*).

Iron containing Si, 0.90; P, 0.20 per cent.
Iron without titanium (average, 5 tests).....42,175 lb.
Iron with 0.5 p. c. of alloy in ladle (average, 3 tests)..44,525 lb., increase 6 p.c.

Using iron which could be treated and poured out without intervening delays—except, of course, the 3-min. wait for treatment—the following results were shown:

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Test No. 12.

Close-grained gray-iron, P more than 1 per cent.

Iron without titanium (average, 4 tests).....35,810 lb.
Iron with 0.20 p. c. of alloy in ladle (average, 4 tests), 39,640 lb., increase 11 p.c.
Iron with 0.64 p. c. of alloy in ladle (average, 4 tests), 40,545 lb., increase 13 p.c.

Had I thought in time to make such arrangements that both treated and untreated iron should be poured at exactly the same temperature, the test would have been still more representative. As it was, the treated iron was poured a little cooler than the other.

Test No. 13.

Gray-Iron.

Iron without titanium (average, 4 tests).....41,520 lb.
Iron with 0.5 p. c. of alloy in ladle (average, 4 tests), 48,350 lb., increase 16 p.c.
Iron with 2 p. c. of alloy in ladle (average, 7 tests), 45,390 lb., increase 9 p.c.

Test No. 14.

White-Iron.

Iron without titanium (average, 2 tests).....32,800 lb.
Iron with 0.5 p. c. of alloy in ladle (average, 2 tests), 37,450 lb., increase 14 p.c.

Two tests were made by adding the alloy in the cupola, the slag being continuously drained:

Test No. 15.

White-Iron.

Iron without titanium (average, 2 tests).....32,800 lb.
Iron with 0.3 p. c. of alloy in cupola (average, 8 tests), 47,040 lb., increase 43 p.c.
Iron with 0.45 p. c. of alloy in cupola (1 test only)....37,600 lb., increase 12 p.c.
Iron with 0.60 p. c. of alloy in cupola (1 test only),[a] 32,950 lb., increase 0.5 p.c.

[a] Last charge in cupola; iron badly oxidized.

Three tests were then made on iron containing between 3 and 4 per cent. of phosphorus, with the following results. The iron was liquid when poured, but not very hot.

Tests Nos. 16, 17, *and* 18.

High-Phosphorus Iron.

Iron without titanium (average, 3 tests).....46,583 lb.
Iron with 0.37 p. c. of alloy in ladle (average, 2 tests)..49,900 lb., increase 7 p.c.

Iron without titanium (average, 2 tests).....47,600 lb.
Iron with 0.5 p. c. of alloy in ladle (1 test only).....54,650 lb., increase 14 p.c.

Iron without titanium.....42,450 lb.
Iron with 1 p.c. of alloy in ladle (average, 2 tests)...46,385 lb., increase 10 p.c.

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Iron containing P about 9 per cent. was tested:

Test No. 19 (*Very High-Phosphorus Iron*).

Iron without titanium (average, 2 tests).....41,900 lb.
Iron with 0.5 p.c. of alloy in ladle (average, 2 tests), 42,630 lb., increase 2 p.c.

The significance of these results is that iron may be increased in strength up to 43 per cent. by the correct use of titanium. However, to accomplish the best results, the titanium must not only be in correct proportions, but the iron must be given a fair chance by allowing the titanium a full opportunity to become thoroughly incorporated in it (not less than three minutes treatment), so that it is neither wasted in the slag nor left unmelted in the ladle, and by insuring that the iron is poured neither too hot nor too cold.

Summary of Results.

A summary of the results obtained by different investigators follows:

		Increased. Tensile Strength Per Cent.	Transverse Strength. Per Cent.
Rossi,	Alloy	30 to 50	20 to 25
Schiemann,	Thermit	30 to 50	
Rossi,	Alloy	6 to 30	17 to 23
Goldschmidt,	Thermit	4 to 16	1 to 17
Moldenke,	Alloy	.....	18 (white)
Moldenke,	Alloy	.....	52 (gray)
Guillier,[91B]	.....	.....	49 (malleable cast-iron)
Guillier,[91B]	.....	.....	14 to 30 (gray)
Feise,	Alloy	.....	35
Treuheit,	Thermit and alloy	No important change.	No important change.
Gale,	Alloy	⅛ to 12	15 to 17[a]
Alexander,	Alloy	(Increased ductility.[a])	
Geiger,	Alloy	13 to 26	6 to 10
West,	Alloy	.....	27 to 32
Stoughton,	Alloy	.....	10 to 43[a]

[a] When treatment was properly performed; otherwise, results different and variable.

These results indicate that it is entirely possible, by correct treatment with ferro-titanium alloy, to increase the strength of cast-iron by from 30 to 50 per cent. If this can be done part of the time, then it only remains to determine the correct conditions in order to attain equal success all the time. Some of

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these conditions I have pointed out in the foregoing pages, but I do not consider that the problem is completely solved as yet. What has been learned is published now in the hope that it will inure to the benefit of makers and users of cast-iron.

APPENDIX.

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DISCUSSION.

J. E. JOHNSON, JR., Ashland, Wis.:—Is it true that titanium reduces the chill in chilling iron? If so, as I think Mr. Stoughton has stated elsewhere, then I do not see how titanium is beneficial in car-wheel mixtures. One of the most difficult things to do is to get a proper chill and a proper degree of toughness, and the goodness of an iron of the given composition for charcoal-iron purposes is denoted by the chill which it will take as compared with its composition. If titanium reduces that chill, then it does not seem to me as good for car-wheels as its absence would be, and I should like to have it explained what effect titanium has on the chill of a given iron.

MR. STOUGHTON: It is true that titanium reduces the chill of iron quite noticeably, yet, at the same time, if I am correctly informed, 90 per cent. of all chilled-iron rolls are made to-day with the titanium treatment, and a great many car-wheels, but one has to make changes in the silicon added and also, if I am

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correctly informed, in the manganese. The advantage of titanium is that when the proper changes are made, so that we get the same depth of chill with the titanium as we otherwise would without, the chill is much harder than before, and also the body of the roll or wheel is tougher and somewhat stronger, especially under shock.

JAMES F. KEMP, New York, N. Y.:—This subject is obviously one of great importance to the iron industry, in that if we can find a method of using titaniferous ores, we will prolong the life of our iron reserves. Fifteen or more years ago, primarily from the study of the ores in the Adirondacks, I became very much interested in this type of deposit.[1] We know that there are many millions of tons of them in that section of the country, and to the north of it in Canada. They are also abundant in Minnesota north of Lake Superior; in Wyoming, Colorado, Virginia and North Carolina, and if we can bring them into actual use they will add greatly to the iron-ore resources of this continent. We also know that in the titaniferous ores vanadium is at home. About fifteen years ago, in platting curves illustrating the composition of the ores, so far as accurate detailed analyses could be obtained, I found that there was a very peculiar sympathy between the curves of chromium and vanadium. I do not think any of us know the form in which vanadium exists in these ores. But the vanadium and the chromium curves went up and down together in very close sympathy. I dare not say that these two elements were combined in a rare mineral, but that there was this sympathy is an undeniable fact. It is quite possible that some new metallurgical improvements will bring within reach of the furnacemen ore that now is avoided, and has been avoided in the past as it it were a plague. At present the titanium is chiefly derived from rutile.

[1] Titaniferous Iron Ores of the Adirondacks, *Nineteenth Annual Report, U. S. Geological Survey*, Pt. III., pp. 377–422 (1899). A Brief Review of the Titaniferous Magnetites, *School of Mines Quarterly*, vol. 20, pp. 323–356; vol. 21, pp. 56–65 (1899).

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The Effect of High Carbon on the Quality of Charcoal-Iron.

BY J. E. JOHNSON, JR., ASHLAND, WIS.

(Cleveland Meeting, October, 1912.)

CHARCOAL-IRON is quantitively so unimportant compared with coke-iron, that its qualitative importance for many industrial purposes is entirely unknown to many coke-furnace-men, and to the great majority of the members of this Institute.

To explain this qualitative importance, and explain some of the factors that affect the quality, are the objects of this paper.

A brief review of the commercial conditions governing the charcoal-iron industry is a necessary preliminary to any adequate treatment of the subject.

The advent of coke-iron, which transformed the whole iron industry commercially, left intact a limited production of charcoal-iron in the East and South, which has been conducted always on the basis of high cost and high quality, with correspondingly high selling-price. In the Northwest, particularly in the Lake Superior district, the development has been along totally different lines, for which the close proximity of the great iron-ore deposits and the great forests of that region were responsible.

When the great white-pine forests were removed there were left behind large areas of hardwood and hemlock, to which little attention was paid in the prosperous years of the white-pine industry. But when the latter declined, the demand for lumber, and the presence of lumber-men, probably more skilled than any others in the world, caused the gradual introduction of these woods into commercial use.

It was a matter of ancient knowledge that hardwood made an excellent charcoal, and to utilize the cordwood left after the saw-logs were removed, by converting it into charcoal and using it to smelt ore from the adjacent mines, was an obvious step.

In some cases the establishment of the furnace occurred in-

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dependently of the lumber-men through common knowledge in the iron trade of the favorable conditions for supply of raw material.

This outline is not intended to be historic in any other sense than as describing broadly the causes which produced a certain condition.

The abundant supply of both iron-ore and hardwood made feasible the establishment of larger plants with greater outputs than prevailed at the Southern and Eastern ones, where there was generally a never-ending; life-and-death struggle for sufficient raw materials of both kinds to keep the operation alive.

Parallel with this development came that of the recovery of the by-products from the production of charcoal: first, methyl alcohol, commonly known as wood-alcohol, and second, acetate of lime, from which are produced acetic acid and acetone. The varnish industry absorbed enormous quantities of both wood-alcohol and acetone, and this consumption steadily grew. This caused the construction of wood-carbonizing plants, of which the so-called by-products became the principal object, and the charcoal the real by-product.

In the palmy days before the repeal of the tax on grain-alcohol used industrially, and before competition had lowered the price of acetate, the by-products would pay for the wood from which they came, its carbonization, and their recovery, leaving the charcoal literally as a by-product.

In order to turn this charcoal into money, some market for it must be had, and the easiest market was to build a furnace, and run it virtually as an adjunct to the chemical plant. In this way many plants were run by men who had an exceedingly limited knowledge of the iron business, and less of the quality of the iron they produced; and as, in their efforts to learn, they had to draw their information principally from the coke-iron trade, the shrine at which they worshipped was high production.

The larger units led, naturally, to the introduction of economies which were not worth while in the similar plants of the East and South, notably the hot blast, from the use of which the coke-iron trade found such great benefits in increasing production and decreasing fuel-consumption. It was natural that the

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charcoal-iron producers should assume that the same conditions would hold with them to the same extent.

The iron was sold for what it would bring, and as the fuel-cost was relatively low, it could naturally be sold at exceedingly low prices compared with Southern and Eastern charcoal-iron, where the wood by-product industry had never taken root successfully in conjunction with the iron business.

Commercial pressure during successive depressions in the iron business took advantage of these conditions and of the actual decline in the quality of Lake Superior charcoal-iron, and forced its price down to a figure ridiculously low, compared with that of the Southern and Eastern irons.

In the summer of 1910, when the Lake Superior Iron & Chemical Co., of New York, was organized and took over six plants in the Lake Superior region, these conditions had prevailed so long that while the price of warm-blast Southern and Eastern irons ranged from $23 to $33 a ton at the furnace, with a ready sale for all that could be made, Lake Superior charcoal-irons of far superior analysis were a drug on the market at from $14 to $16 per ton.

William Wilkins, the General Superintendent of the Iron Department of the Lake Superior Iron & Chemical Co., set himself the problem of finding the reason for this discrepancy in price, and of removing the cause if it was in the actual value of the iron. An investigation to find the conditions affecting quality was, therefore, undertaken at the Ashland plant, of which I was at that time appointed Manager.

An investigation along the lines of consumption showed that different irons have different characters totally independent of their analyses. I do not for a moment intend to imply that the accepted opinions concerning the control of carbon-condition by silicon, fluidity by phosphorus, and chilling-qualities by manganese, are not in a general way correct. I do intend, however, to declare that the presence or absence of these elements, alone, will not account for all, scarcely for a half, of the facts which have long been known, not to us, but to those who used high-grade irons for special purposes.

Investigation showed that there were foundries using chemistry not only for silicons and sulphurs, but also for carbon-determinations and advanced work of that kind. We found

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that such foundries had proved, by the most irrefutable tests, that they could take a certain iron of a given analysis and produce certain results with it in chill-depth, and uniform interlacing of chill with gray, hardness and toughness of chill combined, with corresponding resistance to abrasion, freedom from pits and from local failures of various kinds. With another iron of the same analysis they could not produce these results at all.

This was true not only of charcoal-iron as compared with coke, but of one charcoal-iron and another. We found that some car-wheel manufacturers knew how to produce, and did produce regularly, when they could sell them, wheels that would give service of many thousand miles in excess of that given by other wheels of their own make, made to be sold to people who insisted on having low price rather than quality. We were shown pieces of castings made with these different mixtures which contained ocular evidence sufficient to convince the most obstinate that there were physical variations not dependent on similarity of analyses in any recognized element.

As a result of these conditions there have grown up branches of the foundry trade forming a relatively small fraction of the whole, but of enormous commercial importance nevertheless, whose whole product is based on their being able to secure and to use irons which give them qualities beyond those obtainable from the ordinary coke foundry-irons. These are particularly the manufacturers of chilled rolls and of chilled car-wheels, though there is a growing demand for irons of high strength and close grain, without great chilling-qualities, for many purposes, notably gas-engine cylinders and the like.

The production of chilled car-wheels in America is enormous. The steel wheel has by no means filled the promise implied in the name of "steel." The first-cost is very high and the durability is not great, even with the four expensive turnings which it receives during the course of its life. There is no difficulty whatever in demonstrating that more miles of wheel-service under a given load may be obtained for a dollar with chilled cast-iron wheels made of suitable irons properly mixed, than can be obtained from a steel wheel, and, in fact, probably 90 per cent. of the freight traffic of the country is carried on chilled car-wheels to-day, and will continue to be

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for many years to come. This branch of the business alone is of such vast commercial importance as to justify industrially and financially the great expense which has been incurred by the present investigation.

When the technical investigation was started I had no idea but that within a year or so we would have solved the problems presented to us, of which we had no detailed knowledge at that time. The investigation has now been in progress for two years. In that time we have proved to be true many things which we did not believe, and have proved to be untrue many things of which we were firmly convinced, but we do not yet consider that we are within measurable distance of the final solution of the great problem—why one iron should be so different from another of the same analysis. But experiments on the furnaces themselves, covering the widest possible range at all consistent with commercial operation, have indicated to us methods whereby the quality of the iron may be vastly improved from the state of things described in the introduction.

Some of our investigations on the iron itself have yielded partial results, and careful study of the relation between furnace-operation and quality of iron has enabled us to determine furnace-lines and some of the operating-conditions to give the desired results. Practically all the plants have been rebuilt to embody the results of the investigation to date.

The investigation is still going on, and in its course we have come upon what seems to me the explanation of a phenomenon which has puzzled many—iron with a white or close spot in the center.

The difficulty of the investigation is one that is hard to overestimate, and even more difficult to explain to the satisfaction of those familiar only with the coke-iron trade. I had been in the latter trade practically all my life, and there was no one more deeply immersed in its traditions than myself. I had taken some part in the development of the now-accepted belief that soft steel, hard steel, white iron, and foundry-iron were members of a single series of iron-carbon alloys of increasing carbon-contents, that silicon controlled the condition of the carbon, and that the analysis of pig-iron for the ordinary elements of silicon, sulphur, phosphorus, and manganese gave the final word as to its character.

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The logical ultimate deduction from this theory was that, given the analysis, the character of the iron was established, and virtually no attention need be paid to the fuel used in its production or any other consideration affecting its origin.

It is a matter of common knowledge that the only essential difference in the conditions when manufacturing iron with charcoal instead of coke consists in the fact that the charcoal is so free from sulphur that much less calcareous—and, therefore, more fusible—slags can be used without danger of introducing sulphur into the iron. Any coke-furnace-man, knowing this to be the sole essential difference between the two kinds of iron, and convinced, as most of them are, of the close correspondence between analysis and character, may well find it impossible to believe that such difference can exist, not only in strength but in appearance and structure, between irons of similar analysis, as we actually found in daily practice. That such a widely-variant series of products can exist outside the knowledge of coke-furnace-men is explained by the fact that we could run charcoal-furnaces with slags similar to theirs, and produce a line of products approximately similar to theirs in all respects; but by lowering the lime-content of the slag we can bring into effect a vast series of varying conditions, which are masked, if not altogether obliterated, by basic slags and the high temperatures necessary to fuse them. These conditions are impossible for coke-iron because the resultant rise in its sulphur-content would ruin it. I am moved to lay stress on this point of variations in quality not dependent on variations in analysis by the clear recollection that my own point of view, when I first entered the charcoal-iron field, was that above outlined and probably is that of most furnace-men to-day.

It is true that I had noted with much interest certain variations in coke-iron of the same analysis dependent on varying furnace-conditions, and I had very definite ideas as to the type of fracture that was good, and what was bad, as affected by these conditions. These views, however, were distinctly subordinate to the broad view of analysis control, of which I was a strong advocate.

When we were confronted with the overwhelming commercial fact that the Southern and Eastern irons sold for from 50 to 100 per cent. more than we could obtain for our product,

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the natural and obvious thing to do was to claim ignorance and prejudice on the part of the foundry-men, and poor salesmanship on the part of the agents, because any tyro could compare the analyses of our iron with that of the Southern and Eastern irons and see that ours were the better.

Gradually, however, the indefatigable efforts of Mr. Wilkins brought us into contact with foundry-men who obviously knew their business at least as thoroughly in both practice and theory as we knew ours, and they gradually convinced us by the hard logic of irrefutable physical facts that the difference in price corresponded to a difference in quality and in ignorance, as we had supposed, but that we, and not they, were the possessors of the surplus stock of ignorance.

Experience with these Lake Superior charcoal-irons, taken in conjunction with frequent analyses, had led to the adoption of a grading-card, which is given herewith:

Grade.	Silicon, Average.	Chill Against Plate.	Manganese. Car-Wheel Grade.	Manganese. Malleable Grade.
	Per Cent.		Per Cent.	Per Cent.
AA, Scotch	3.00 and over.	.....	.....	.....
A, Scotch	2.75 and over.	.....	.....	.....
B, Scotch	2.50 and over.	.....	.....	.....
C, Scotch	2.25 and over.	.....	.....	.....
No. 1, Soft	2.00 and over.	.....	0.70 or over.	Under 0.70
No. 1, Special	1.75 and over.	.....	0.70 or over.	Under 0.70
No. 1, Foundry	1.50 and over.	.....	0.70 or over.	Under 0.70
No. 2, Low	1.25 and over.	.....	0.70 or over.	Under 0.70
No. 2, High	1.00 and over.	No chill.	0.70 or over.	Under 0.70
No. 3, Low	0.80 and over.	Feather.	0.60 or over.	Under 0.60
No. 3, Med	0.65 and over.	⅛ to ¾ in.	0.55 or over.	Under 0.55
No. 3, High	0.55 and over.	⅝ to ¾ in.	0.50 or over.	Under 0.50
No. 4, Low	0.45 and over.	½ to 1¼ in.	0.45 or over.	Under 0.45
No. 4, High	0.35 and over.	1¼ to 2 in.	0.40 or over.	Under 0.40
No. 5, Low	0.30 and over.	Near white.	0.35 or over.	Under 0.35
No. 5, High	0.20 and over.	White, spot of graphite.	0.30 or over.	Under 0.30
No. 6, White	0.10 and over.	Clear White.	.....	.....

It will be noticed that there is an assigned depth of chill for each analysis. To gauge the chilling-qualities of an iron it is the custom to have a heavy block of cast-iron planed on one side, which forms a portion of one side of the pig-mold for one pig in every bed. The chill on this chill-block is the one referred to in the grading-card given above.

The grading-card was based on the three principal lines of consumption of these irons, as follows:

1. Straight foundry-work, especially that of the better class, where strength and density of grain were required, but not chilling-qualities.

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2. The production of chilled castings, principally car-wheels and chilled rolls, where manganese was supposed to be beneficial.

3. The production of malleable cast-iron, in which low manganese is generally specified.

It will be noticed that the silicon-limits from one grade to another are very narrow, and that the grades run much higher than No. 1. This is probably, to some extent, because the small quantity of sulphur present does not neutralize much of the effect of the silicon, so that the latter reaches the limit of its effect much earlier than in irons with higher sulphur, so consequently No. 1 soft charcoal, with 2 per cent. of silicon, corresponds in some of its qualities to No. 1 soft coke-iron, with 3.25 per cent. of silicon.

For the foundry trade silicon is needed for scrap-carrying purposes and the like, as with coke-irons, and irons with more silicon than No. 1 soft are graded as Scotch irons.

Because of the rapid variation in grade for small variations in silicon, the practice was early established of taking a sample for silicon-analysis from every third bed, and grading the iron by the results of these silicon-determinations. Two manganese-determinations were also made on each cast. This grading-card was established several years ago, and, while it has recently been amplified, the silicon-limits of the grades are practically unchanged.

When we began our investigation one of the great difficulties with which we had to contend was to find out what was good iron, and what was bad. Was a strong iron necessarily a good iron, or a weak iron necessarily a bad one? It will seem inconceivable, but it took a year and one-half before we found any one to give us a definite answer to this question. It was finally answered by the superintendent of the car-wheel foundry of a large railroad system, who, by his practical knowledge of the business, and by physical tests of the irons he purchased, combined with judicious mixing and excellent foundry-practice, produced wheels that stood an average of 95,000 miles in service, as against an average of from 30,000 to 45,000 miles for the wheels of the ordinary commercial wheel-foundries.

This gentleman assured us that for his purpose a good iron

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was a strong one, and a strong one a good one. The standard of strength with him was a 2-in. square bar, broken transversely on centers 12 in. apart. If the strength for a given grade was up to certain limits, and the chill of a certain depth, the iron was good. If not, the iron was undesirable for his purposes and was rejected.

This condition of complete ignorance as to what to investigate caused the investigation to take shape very slowly. We started at once with pyrometric observation of the temperature of the running iron and cinder, and bought a small transverse testing-machine, capable of breaking a 1.25-in. round bar on 12-in. centers, the standard recommended by the American Foundrymen's Association. We cast four bars on end in green-sand molds with iron dipped directly from the runner, at each cast, and broke these on the testing-machine. We believed, however, that we should have a record of the stress-strain diagram of the test-bars and we accordingly bought, a little later, an autographic machine, designed and made by Tinius Olsen & Co., of Philadelphia, for this purpose. This machine draws a line on a 5-in. square chart, the abscissæ of the line showing the stress applied, and ordinates, the corresponding deflection, right up to the point of rupture. After certain minor changes were made this machine was easy to manipulate, and the quality of the iron was soon found to have a decided bearing on the character of the curves produced.

There were two well-established traditions in the business when the investigation was begun. One was that an iron with a white chilled spot in the center of the pig, which is sometimes produced, was weak and utterly worthless for the manufacture of chilled car-wheels, or in any other purpose requiring a strong, tough chill. This is commonly known as "spotted iron." Second, that an iron was made under certain conditions in which the grain was exceedingly close, verging slowly into a chill of greater intensity than the silicon-content of the iron would explain, and that this was a very strong, tough iron, particularly suited for cylinder-castings on account of its high strength and exceedingly close grain, taking a high finish. In consequence it was given an individual name, "special cylinder-iron," and saved for certain customers to whom it was particularly valuable, though it is a lamentable fact they formerly

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got it without paying anything like an adequate price for it. This iron is produced as the result of certain unusual furnace-conditions, and may have any silicon-content from 0.5 to 2 per cent.

The first positive results of the investigation were to prove that this special cylinder-iron was far stronger than normal, and that, on the other hand, the spotted iron was far weaker, thereby confirming thoroughly the traditions of the practical men, whom it had been the custom to deride.

At the expense of much time and money, Mr. Wilkins secured samples of the best brands of Southern and Eastern irons, from which remelts were made in order to determine their strength, also complete chemical analyses for comparison with our own. Gradually it developed that we needed something more, and a microscopic outfit and polishing-apparatus were added. Subsequently it became a conviction in the minds of all, particularly in that of Mr. Wilkins, that reliable carbon-determinations were a necessity.

It is well known to all who have tried to get carbon-determinations of pig-iron that there is no greater bugbear in the laboratory, and the attempt to obtain accurate ones lasted for a year before we had discovered all the conditions necessary to get reliable results. We soon found, however, that one carbon-determination per day was not enough, and we then added an extra man in the laboratory and began to get total and graphitic carbons on every cast, in addition to the six silicon- and two manganese-determinations for each cast. We added also a phosphorus- and a sulphur-determination for each cast, and by means of the latter, destroyed the beautiful delusion that good charcoal-iron ordinarily contains about 0.01 per cent. of sulphur, since accurate determinations show nearly all of it to contain from 0.015 to 0.020 per cent., even when made from the best of ores, and very much higher if there is more than a trace of sulphur in the ore.

More than a year ago we put in a crucible-furnace for making crucible-remelts of irons, to determine the effect of remelting on our own iron, and to obtain results comparative with those of others which are generally made on remelted iron. If the test is to represent the actual quality of the iron, crucible-remelts must be used to prevent oxidation, contamination with sulphur, and the like.

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We at first made these remelts only on very unusual irons of our own, or on irons from outside sources, and used the 1.25-in. round test-bar; but later, finding what great dependence the foundry-men placed on the 2-in. square bar, broken transversely on 12-in. centers, we began making remelts of every cast of iron suitable for chilling purposes, that is, everything below the No. 1 grades, making both 2-in. and 1-in. square test-bars.[1]

In order to utilize all this information, the iron is all piled by grades, but the portion of a given pile which comes from a given cast is bulkheaded off from the portions on either side of it and marked with a tag, giving the date and hour of the cast and the silicon and manganese of that portion of the pile. The daily furnace-reports carry the ordinary information contained in a daily report, and, in addition, the carbon-determinations, at least one slag-analysis, the temperature of the running iron and cinder, the breaking-strength of 1.25-in. round bars poured from the cast, and the breaking-strength of the remelt-bars on the chilling grades of iron. By means of this system we know the exact iron which goes out on each order, and when we get an order for exactly the same kind of iron, we can take our records and locate it with little or no difficulty. In this way we are able to direct the various grades and kinds of iron to the fields of consumption to which they are best adapted. This system involves extensive records and a considerable amount of clerical work, but gives results so satisfactory to consumers that we find it well worth while.

[1] The transverse strength of a good 2-in. square test-bar on 12-in. centers will sometimes go as high as 30,000 lb., and to break these and at the same time avoid the expense of buying a testing-machine of this capacity, we rigged up a removable girder over our track-scale to carry a screw turned by a ratchet-lever, strong enough for the purpose. The screw carries at its lower end a knife-edge, which is guided by a square shank to prevent its turning. The test-bar is placed on a pair of knife-edges, 12 in. apart, resting on transverse I-beams inserted between the main I-beams of the scale-platform. Guide-pins run through guides mounted on the girder which carries the screw and through guide-holes in the plate carrying bottom knife-edges. These place the bottom knife-edges accurately under the center of the top knife-edge in both directions. The stress is put on the bar by the hand-lever, as shown, and the load on the bar weighed on the scale-beam. With very little practice it is possible to get results accurate within 100 lb. or less. The apparatus, a view of which is given in Fig. 1, serves the purpose of an expensive testing-machine very satisfactorily for those who have a relatively small amount of such work to do.

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The opinion gradually took shape in my mind that the spotted iron, the normal iron, and the special cylinder-iron were the members of a continuous series of increasing quality, and that if we could find out the cause of the spotted iron and the proper method of preventing its production, we would have at least part of the problem solved. This view has gradually been confirmed by the progress of the investigation, and we have further found some of the factors which tend to increase the strength of the iron and move it from the normal to the special cylinder quality.

We do not feel that we have yet reached the complete solution of these matters in such a degree as would make their publication safe and desirable, but we think we have succeeded in finding the cause of the spotted iron and in connecting up its analysis and physical qualities and assigning to it its proper place on the iron-carbon diagram.

Fig. 2 is a photograph of a representative piece of spotted iron. It shows the white spot in the center with two rudimentary branches extending towards the upper corners of the pig. When I first gave thorough consideration to this kind of iron it seemed to me that the location of the white spot indicated that it was a segregation-product, and I had analyses made of the gray portion outside and of the white spot, with the result of finding that the total carbon was about 0.5 per cent. higher in the former than in the latter, but no material variation in the other elements, certainly none in either the silicon or the manganese, which would account for the internal chill in spite of its being slowly cooled.

I was informed by experts in the metallurgy of iron that the ordinary explanation of this phenomenon was that the center of the pig was still liquid when the exterior had solidified, and that the expansion which accompanies the evolution of graphite was not permitted to take place on account of the contraction of the solid envelope on the liquid interior; but as iron of all grades cools under precisely these conditions, and only a small percentage of it is spotted, this explanation could not be regarded as satisfactory.

Gradually investigation and observation developed the fact that the irons in the low and high No. 4 grades were most apt to be spotted, but that a tendency to spot was sometimes

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shown in softer grades, notably the No. 3 iron, even though the pig itself might not show any trace of spot. For instance, the test-pins in these grades might have a very small spot, even when the pig had none at all.

We have repeatedly heard the term "high cleavage" used in connection with the Lake Superior charcoal-irons by experienced users, and we had at one time considered that this was part of their unreasonable prejudice against this iron, but we gradually found that certain irons which had this tendency to spot showed a degree of lamination where they were chilled, or partly chilled, that was almost beyond belief. In the corners of some pigs there could be seen with the naked eye alternate layers, perfectly white and solid gray, the white leaves being no thicker than ordinary writing-paper, the gray perhaps twice as thick. It was obvious that such iron must break along the faces of these leaves of white, and that the term "high cleavage" expressed its quality more accurately than any other equally simple term.

Observation gradually brought out the fact that while the special cylinder-irons showed a tendency to chill much more than their silicon-content would lead one to expect, those irons with a tendency to spot, even though not actually spotted, showed a chill much lower than their analysis would warrant, and more than a year ago we began the practice of marking such iron with a distinctive mark and forbidding its shipment to manufacturers of chilled car-wheels or any similar article, because we found the testing-machine confirmed the ocular judgment that these irons were of the general nature of spotted iron, and therefore very weak. We had photomicrographs made of the iron shown in Fig. 2, particularly of the spotted portions, which are reproduced in Figs. 3, 4, and 5. It is obvious that the masses of detached plates of which these areas are made up simply abut against one another and are not in any way interlaced, so that a very low intensity of stress will cause cleavage along the face of one or the other of them. We finally recognized the structure as shown by the photographs to be that of eutectic cast-iron of 4.30 per cent. of carbon, as shown by Fig. 118 of Goeren's *Introduction to Metallurgy*, p. 175. Simultaneously with the microscopic examination of a specimen of this iron we repeated our earlier determinations

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of the carbon in the light of the most recent developments in these determinations, with a view to corroborating our earlier results, and we obtained the results shown in Fig. 8. It then became evident that our earlier opinion that this was a segregation-product was correct, and that the white chilled spot was the eutectic of 4.30 per cent. of carbon, which had gone to the same relative position in the pig as that in which we find the segregated portion of a steel ingot, and for the same reason: that both had remained molten on account of their low freezing-temperature after all the other portions of the mass were solidified.

From this point the investigation and this description of it naturally divides into two parts. First, what are the causes of this action? Second, how far does the action extend and how serious are its effects in industrial work?

Without any desire, or ability, to make this a purely scientific matter, it was necessary to align the results of the investigation with the established iron-carbon diagram in order to check our results, and the portion of the article from this point to the end of the second paragraph on page 344, seems to us the explanation of the observed facts on this basis. From that point on the conclusions based upon those facts are given. My rudimentary knowledge of the iron-carbon diagram prevents the former portion from being particularly technical, but it may be undesirably so, for those who desire conclusions only, and all such may omit the former portion without other loss than the danger of being misled by erroneous conclusions in the latter section. As these are based principally on observation, and but little on deduction, this danger is not serious.

It so happened that our first investigation of carbon-content in various portions of the face of the pig was made not on the iron shown in Fig. 2, but on that shown in Fig. 6. These carbon-determination and their position are shown by Fig. 7. It will be observed that the carbons in the gray portion are all much higher than that in the spot, and being unfamiliar with the interpretation of the iron-carbon diagram we first assumed that it was only hyper-euctectic irons which could thus segregate, and that the phenomenon was one confined to the right-hand branch of the iron-carbon liquid diagram.

This error was fostered by the fact that we had a suspicion

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that the high-carbon irons were the bad ones. We came gradually to realize, however, that this explanation was not valid, and that segregation of eutectic could take place in hypo-eutectic irons as well as in hyper-eutectic, so that any interpretation of the facts based on the former assumption only was wrong.

It is obvious that irons on the right-hand side of the eutectic point throw out only graphite in cooling down to the eutectic freezing-point, so that the ground-mass of the whole pig, exclusive of the small proportion of graphite so thrown out, is eutectic at the time of final solidification. But the center is free of these particles of this graphite because it has remained liquid so much longer as to give them a chance to separate out by gravity into the portion above the spot. Therefore, when the iron cools down below the eutectic freezing-point, the portion around the spot contains many particles of graphite which probably act as foci to cause other graphite to precipitate out of the mass from the solid condition. The center is without these foci, and is moreover under compression from the contraction of the previously solidified exterior, which prevents the expansion necessary for the evolution of graphite, therefore no graphite forms.

The iron shown in Fig. 2 was subsequently analyzed for carbon at eight points, as shown in Fig. 8. It will be seen that the spot contains the highest carbon, except the gray portion immediately above it, which is only slightly higher. The first-cooled portions of the pig, however, are all lower in carbon than the spot. In these portions solidification has obviously taken place, first, as austenite, with a lower carbon-content and correspondingly higher freezing-point than the eutectic. These portions had no deposition of foci of graphite while liquid to induce the precipitation of more graphite during cooling in the solid state, but had the ability to form a solid structure of austenite (undoubtedly immeshing pools of eutectic therein), which would solidify earlier, and, therefore, be in better position to exert pressure during contraction on the central liquid portion than if all froze at the same temperature. In other words, the austenite is capable of making, and does make, a continuous structure with a freezing-point higher than the eutectic, whereas, the graphite in the hyper-eutectic irons cannot form

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any continuous structure, and does not affect the freezing-temperature of the principal portion of the exterior mass.

We have here also the obvious action that when there is carbon in excess of the eutectic ratio, the excess falls out, and the whole mass, except these isolated flakes of primary graphite, is composed of eutectic. In the hypo-eutectic irons, on the other hand, it is obvious that less eutectic can be formed according as the carbon-content of the whole is further below the eutectic ratio.

In the hyper-eutectic irons the isolated particles of graphite can exercise but little influence on the formation of the plates of eutectic, whose shape, size, and utter lack of interlacing ac-

FIG. 1.—IMPROVISED TRANSVERSE-TESTING APPARATUS.

count for the poor physical qualities of this kind of iron. In the hypo-eutectic irons, on the other hand, a skeleton of austenite, called by Professor Howe primaustenoid, forms first, and, being relatively solid by being well below its freezing point at the time of the freezing of the eutectic, can prevent the formation of large plates of the latter, which, if formed, act on the mass as a whole like gigantic cleavage-planes. The further the carbon-content of the whole is below the eutectic ratio, the earlier must be the freezing of the austenite, and the greater its quantity before the formation of the eutectic can begin. Therefore, the more powerful will it be in preventing

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the formation of the huge plates of eutectic when the eutectic freezing-point is reached, while the quantity of the latter which can form is diminished at a still more rapid rate.

If this explanation be correct it is obvious that the cleavage condition, visible to the naked eye, must be vastly more extensive in a hyper-eutectic spotted iron than in a hypo-eutectic one, and that it must extend to the very exterior of the pig in the former, and be more and more restricted to the center as

FIG. 2.—TYPICAL SPOTTED IRON (No. 4).

the carbon is lower in the latter. These conditions we find in fact to exist. The fracture of the iron at the exterior of the pig shown in Fig. 2 is not particularly high cleavage, whereas, the iron shown in Fig. 6 is of most peculiar appearance. There are close-grained spots scattered all over its face, not perfectly white, but visible to the naked eye upon close examination as plates of solid white alternating with layers of gray; in other

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FIG. 3.—WHITE PORTION OF FIG. 2. ETCHED. MAGNIFIED 70 DIAMETERS.

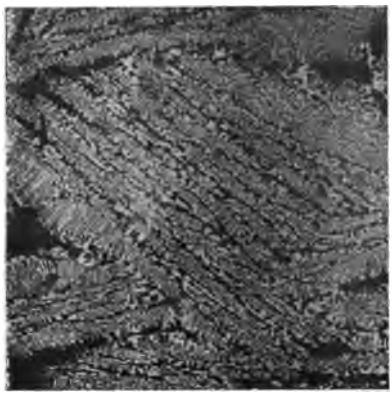

FIG. 4.—WHITE PORTION OF FIG. 2. ETCHED. MAGNIFIED 38 DIAMETERS.

FIG. 5.—WHITE PORTION OF FIG. 2. ETCHED. MAGNIFIED 100 DIAMETERS.

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FIG. 6.—LOW NO. 4 SPOTTED IRON. VERY HIGH CLEAVAGE.

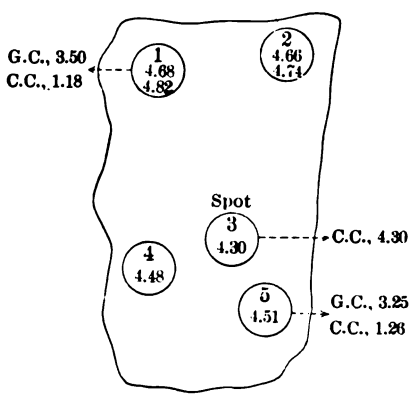

FIG. 7.—CARBON-DISTRIBUTION IN FIG. 6.

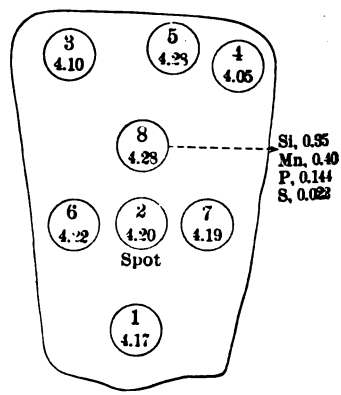

FIG. 8.—CARBON-DISTRIBUTION IN FIG. 2.

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FIG. 9.—CORNER OF PIG SHOWN IN FIG. 6, SHOWING LAMINATION. UNETCHED. MAGNIFIED 70 DIAMETERS.

FIG. 10.—CORNER OF PIG SHOWN IN FIG. 6, SHOWING LAMINATION. UNETCHED. MAGNIFIED 70 DIAMETERS.

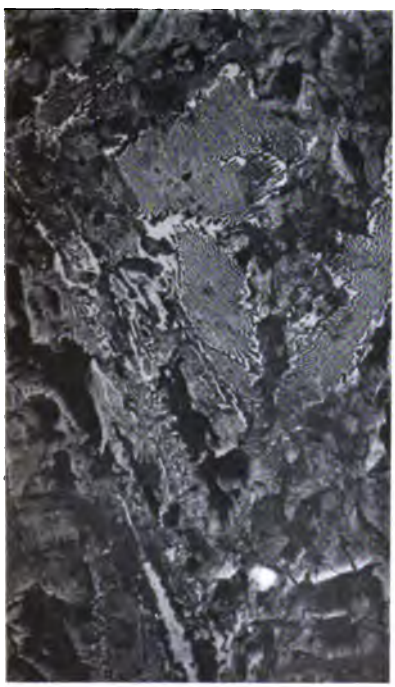

FIG. 11.—CORNER OF PIG SHOWN IN FIG. 6, SHOWING LAMINATION. ETCHED. MAGNIFIED 70 DIAMETERS.

FIG. 12.—CORNER OF PIG SHOWN IN FIG. 6, SHOWING LAMINATION. ETCHED. MAGNIFIED 450 DIAMETERS.

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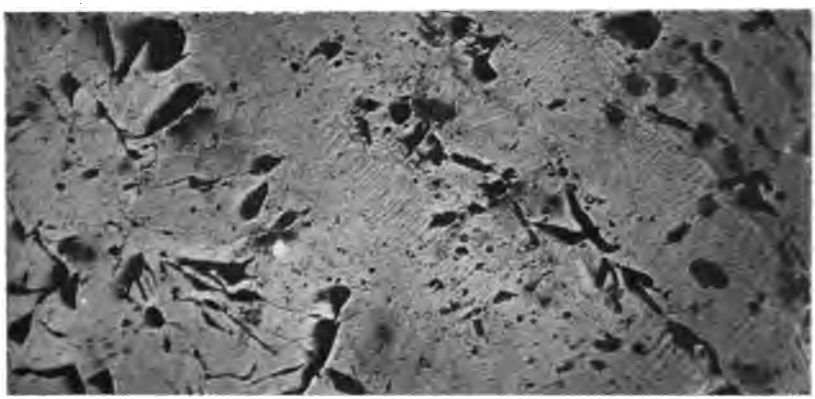

FIG. 13.—CORNER OF PIG SHOWN IN FIG. 2. NOTE, RELATIVELY SLIGHT LAMINATION COMPARED WITH FIGS. 9 AND 10. UNETCHED. MAGNIFIED 100 DIAMETERS.

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FIG. 14.—HIGH NO. 3 IRON WITH TENDENCY TO SPOT.

FIG. 15.—CORNER OF PIG SHOWN IN FIG. 14. UNETCHED. MAGNIFIED 70 DIAMETERS.

FIG. 16.—CORNER OF PIG SHOWN IN FIG. 14. ETCHED. MAGNIFIED 70 DIAMETERS.

FIG. 17.—CORNER OF PIG SHOWN IN FIG. 14. ETCHED. MAGNIFIED 450 DIAMETERS.

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FIG. 18.—LOW NO. 2 IRON WITH TENDENCY TO SPOT.

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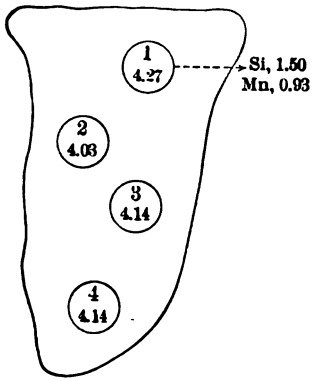

FIG. 19.—CARBON-DISTRIBUTION IN FIG. 18.

FIG. 20.—UPPER CORNER OF PIG SHOWN IN FIG. 18. NOTE ORIENTATION OF GRAPHITE. UNETCHED. MAGNIFIED 38 DIAMETERS.

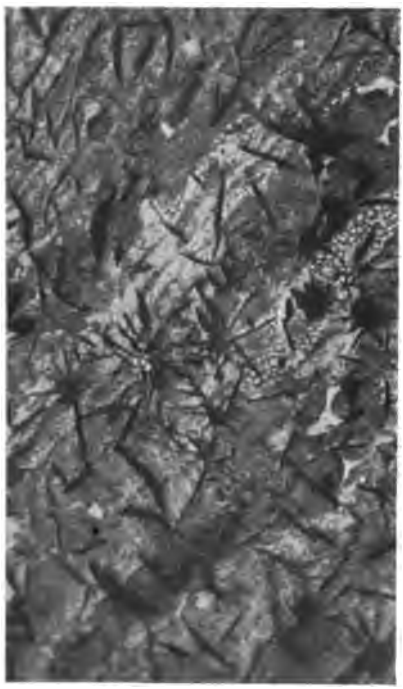

FIG. 21.—CORNER OF PIG SHOWN IN FIG. 18. ETCHED. MAGNIFIED 100 DIAMETERS.

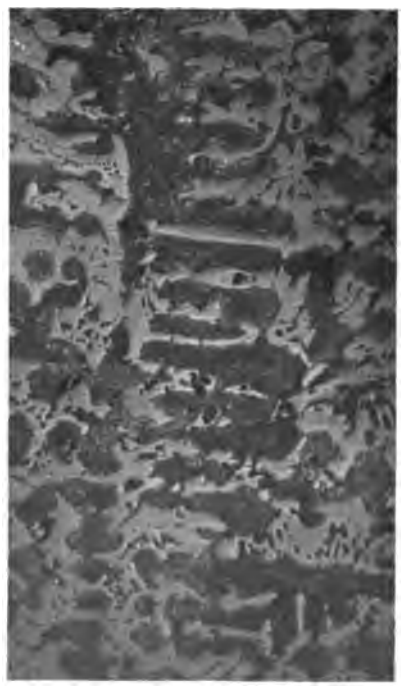

FIG. 22.—WHITE IRON OF 3.71 PER CENT. OF CARBON. ETCHED. MAGNIFIED 100 DIAMETERS.

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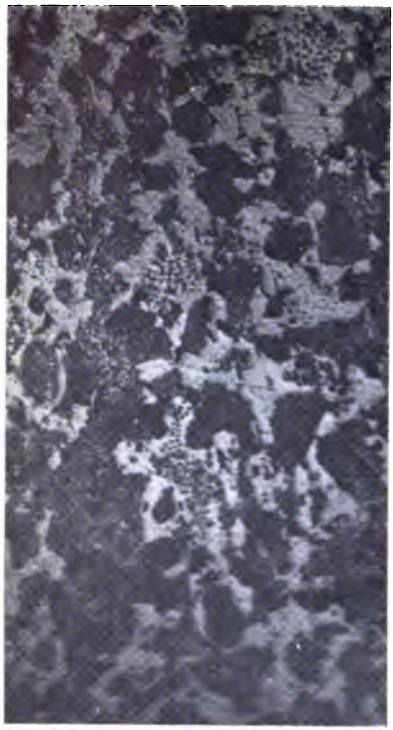

FIG. 23.—WHITE IRON OF 3.30 PER CENT. OF CARBON. ETCHED. MAGNIFIED 100 DIAMETERS.

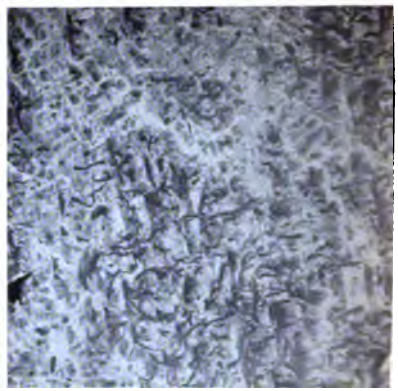

FIG. 24.—NEWBERRY HIGH NO. 3 IRON. UPPER CORNER OF PIG. 3.50 PER CENT. OF TOTAL CARBON. ETCHED. MAGNIFIED 38 DIAMETERS.

FIG. 25.—NEWBERRY LOW-CARBON IRON. ETCHED. MAGNIFIED 100 DIAMETERS.

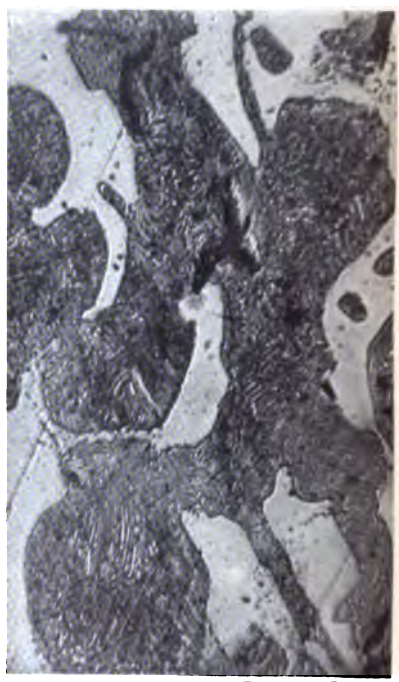

FIG. 26.—NEWBERRY LOW-CARBON IRON. ETCHED. MAGNIFIED 450 DIAMETERS.

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words, the most accentuated condition of cleavage we could find. This cleavage condition is usually manifested particularly in the upper corners of the pig, and from one of these corners of this pig we took a specimen, which was polished and photographed without etching, so as to show the lamination and orientation of the graphite. This is shown in Figs. 9 and 10. The same sample, etched and at the same magnification, is shown in Fig. 11, and under higher magnification in Fig. 12.

In order to prove if a contrast existed, we took a similar corner of the pig shown in Fig. 2 and prepared a specimen in the same way. The result is shown in Fig. 13, unetched.

It will be seen at a glance that the laminated condition is much less and the graphite much less oriented in this specimen than in the other. This iron being a hypo-eutectic iron the portion which froze first was austenite, and there were only puddles of eutectic left to make the few long flakes of this material shown. The graphite is fine, scattered and nodular, an indication of its being secondary graphite, while some of the graphite in Figs. 9 and 10 is large and in relatively long flakes, an indication of its being primary.

Allusion has been made to irons which have a tendency to spot without being actually spotted. These irons usually occur in the No. 3 grades. In addition to these, we have noticed irons in the No. 2 and even softer grades with a close spot in the center, surrounded by more open-grained material.

After we had this clue it was relatively easy to find from the results of the test-bars that those casts with this fracture were likely to be below the normal strength for their grades, and by examining the upper corner of the pig we were able, in some cases, to detect slight traces of cleavage. We accordingly prepared a specimen from the upper corner of a pig of high No. 3 iron and photographed it without etching.

The original pig, containing, Si, 0.45; Mn, 0.38; P, 0.124; S, 0.019, and total C, 4.46 per cent., is shown in Fig. 14, and the photomicrograph of the specimen in Fig. 15. The cleavage-planes and the orientation of the rows of graphite show a structure obviously identical with that in Figs. 9 and 10, though slightly less pronounced. This also was a hyper-eutectic iron, as shown by its analysis. The same structure, etched, is shown in Fig. 16.

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If the theory outlined above is correct it is obvious some of the graphite in the hyper-eutectic irons is primary, and if so, it will naturally occur in relatively large flakes quite thin in relation to their length and breadth. Moreover, the graphite, not having any continuity of structure, is obviously at the mercy of the eutectic crystals, and when the latter begin to form they can push the graphite flakes around so that while the long axes of the flakes may not be aligned, the flakes as a body are arranged in "windrows." This condition is plainly evident in Figs. 9, 10, 11, 12, 15, and 16. Fig. 17 is the same as Fig. 16 at a higher magnification and shows the same condition within its limited field.

It will be noticed that the plates of eutectic shown in Figs. 15, 16, and 17 are considerably more broken up than the similar areas in Figs. 3 and 4. This I take to be the result of the greater silicon-content of pig shown in Fig. 14 than of that shown in Fig. 2, and is reflected in the solid gray fracture of the latter as compared with the partly-white fracture of the former. The effect of silicon is obviously to break up these plates of eutectic into pearlite and cementite, and then, in some cases, to break up the latter into pearlite and graphite. If this be true, then we should have an orientation of the graphite even in irons so high in silicon as to have left scarcely any traces of the original eutectic.

In order to prove or disprove this hypothesis we took a pig of low No. 2, shown in Fig. 18, in the corners of which we found indications of cleavage, and prepared a specimen from its upper corner, which is shown in its unetched condition under low magnification in Fig. 20. The orientation of the graphite from the upper left- to the lower right-hand corner is plainly visible. The same specimen etched is shown under a higher magnification in Fig. 21. In this the long windrows of white dots represent all that is left of the cementite, which once composed part of the eutectic plates. The darker rows of graphite flakes are plainly observable between the windrows of white dots. The carbon-determinations taken in the face of this pig are shown in Fig. 19.

We seem justified in believing that this was a hyper-eutectic iron in which the silicon had forced the eutectic point down to about 4 per cent. instead of 4.30 per cent. The fact that the

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segregated spot is lower in carbon than the rest of the pig, and the presence of the residual plates of eutectic in the corners, which we have seen are most likely in hyper-eutectic irons, are our justification for this belief.

In order to confirm the above theory more fully, we took one normal white iron containing 3.71 per cent. of carbon, made under ordinary working-conditions for white iron, and a white iron made on a very cold furnace, containing only 3.3 per cent. of carbon, and prepared specimens from each. Photomicrographs of these are shown in Figs. 22 and 23. The former will be seen to have a well-developed pine-tree structure at the point photographed, and at other points in the same sample this structure was even more markedly developed. In the low-carbon iron it will be seen that there is no sign of the pine-tree structure, and careful search of the whole field of the specimen reveals scarcely any of it, thus proving that when the iron is far enough below the eutectic ratio this structure cannot form to any appreciable extent.

The fact that eutectic can form in iron almost free from silicon, as low in carbon as 3.71 per cent. and practically without segregation (the carbon in the center being 3.72 as against 3.71 per cent. at the side), may be thought contradictory to the theory here advanced. Examination of this iron under the microscope seems to indicate that there probably was the early formation of austenite, which had its effect in orienting the flat crystals of eutectic as they formed, so that they are much more interlaced than any higher carbon white areas. It would seem as though the original austenitic structure had been overwhelmed by the crystals of eutectic freezing from the pools of that material inclosed by the austenite network.

While these investigations were in progress R. W. Durrett, Manager of the Newberry plant of the Lake Superior Iron & Chemical Co., determined some of the conditions controlling the carbon-content of pig-iron and was enabled to produce irons of very low carbon, in some cases as low as 3.3 per cent. A specimen of this iron, having an analysis, Si, 0.55; Mn, 0.18; P, 0.260; S, 0.042; total C, 3.5 per cent., on the closest examination by eye failed to show any trace of cleavage and there was not the slightest indication of spotting. Specimens

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of the iron were prepared for the microscope, and photographs of the results under different magnifications are shown in Figs. 24, 25, and 26. The irregular polygons of primaustenoid are quite plainly visible, but there is not the slightest sign of any residual of the eutectic. This iron is evidently so far below the eutectic ratio that the heavy network of primaustenoid completely overwhelms the attempt of the eutectic to crystallize into flat plates. This is the easier as but very little of the eutectic can form with this low carbon-content. The structure of primaustenoid is one which we have learned to associate with high strength, and there is no doubt that a test of the iron would show a high result, though we have not such a test available.

We have endeavored to obtain specimens of coke-iron containing such white spots in the center, but so far without success. My recollection is quite distinct, however, that these irons are not infrequent in basic practice. We shall take the first opportunity to determine whether the same conditions hold in those irons as in those here shown.

Allusion has been made to the fact that irons, either spotted or with a tendency to spot, are commonly deficient in chilling-qualities as well as strength, and that such irons are specially marked when made, and never shipped to consumers who require those qualities.

The reasons why irons with a tendency to spot are deficient in chilling-qualities is a point which we have not yet worked out. We have established beyond the possibility of a doubt that it is not dependent solely on the carbon in iron. We have seen two irons of the low No. 4 grade of which the carbons were identical within the limits of chemical error (4.06 and 4.07 per cent.), and yet the chill on one was about 0.25 in. deep, on the other about 0.75 in. deep.

It is entirely possible that part of the cause may be the temperature at which the iron is cast. The tendency is, in our opinion, for the chill to be reduced by casting at a higher temperature. It is fairly well established that high hearth-temperature (other things being equal) tends to promote high carbon. It may, therefore, be that deficient chill and high carbon are the joint results of the same cause, high hearth-temperature; rather than having any relation of cause and effect with one

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another. It must be borne in mind, however, that, as stated in the beginning, there is some other factor, perhaps more than one, affecting the quality of the iron in addition to analysis in the ordinary elements, even including the results herein contained concerning carbon. We have had this fact forced on our attention in so many ways that we feel called on to reiterate it at every opportunity, but I do not wish to go into that subject now. It would take me far beyond the legitimate bounds of the present article.

It is not at first sight obvious why these spotted irons should be confined, roughly, within silicon-limits of 0.25 and 0.75 per cent. The reason is probably twofold. Very little is known about the conditions which control the amount of carbon in pig-iron, but two of the facts well understood are that a cold hearth producing white iron gives a low carbon. On the other hand, a hot furnace giving an iron high in silicon also produces a low carbon because of the tendency which carbon and silicon mutually exert towards excluding one another from iron within certain limits. The common illustration of this is "silvery" iron, in which the silicon is so high as to keep the amount of carbon absorbed well below the ordinary range, leaving the white of the iron to show through more than the mass of graphite in normal irons will permit. Between these two extremes of low silicon and high silicon the carbon-content of the iron rises to a maximum, presumably around 0.5 per cent. in silicon, the hearth not being cold enough when running on such iron to prevent the absorption of carbon on one hand and the silicon not high enough to crowd it out, on the other.

These irons, therefore, under ordinary conditions are the most likely of any to contain carbon well above the eutectic point and so to lend themselves to segregation and the consequent formation of the austenite-pearlite eutectic. Moreover, these irons contain enough silicon to throw some of the carbon into the graphitic form as they cool below the eutectic temperature, but they do not contain enough to throw out graphite from the segregated spot against the compression of the solid exterior.

When the silicon is lower than this its effect is insufficient to throw any marked amount of carbon into the form of graphite, below the eutectic freezing-point, even though the iron

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throws out a little primary graphite while cooling down to that point. Irons higher in silicon, on the other hand, exert so great a graphite-forming tendency as to cause its formation from the solid in the segregated spot, but on account of resistance to its formation the graphite tends to take the nodular form, occupying the least possible room, rather than to take its customary flake form. This causes the fine-grained gray structure in the segregated spot.

In regard to these comparatively soft irons with a close spot in the center, it is necessary to exercise great caution in passing judgment. The primitive eutectic plates are so completely broken up that graphite cuts through them as well as parallels them. Moreover, during the process of breaking down, these flat plates form ribs at right angles to the principal axis of the crystal, and these ribs give an interlacing effect which is entirely wanting in the partly-white spotted irons. There is much reason, both practical and theoretical, to believe that some very excellent irons may contain a close spot in the center.

Conclusions.

1. Spotted irons are products of carbon-segregation.

2. This segregation may take place in irons containing either more or less carbon than the eutectic ratio.

3. The segregation is less marked and less likely to have ill effects the further the carbon is below the eutectic ratio.

4. When the fracture of an iron shows high cleavage at the edge it is almost sure to be up to, or above, the eutectic ratio, and is virtually certain to be a weak iron.

5. Close spots not associated with cleavage may occur in irons of good quality, especially where the silicon is comparatively high.

6. While we have discovered the cause of spotted irons and irons of high cleavage, and the reason for their corresponding weakness, it is my firm conviction that there is another condition, element, substance, or what you will, that makes for strength and for high chill with a given silicon-content, in just the same way that high carbon is here shown to make for weakness and low chill with a given silicon-content. What this unknown condition may be I do not yet know, though I have some unconfirmed opinions on the subject. But there

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are many things which make us believe that with this other condition properly fulfilled, a relatively high carbon iron may be stronger and better than an iron of much lower carbon, with this condition absent.

In illustration of this it may be well to call attention to cold-blast irons, which command a price much higher than that of any other kind of pig-iron, and which our investigations nevertheless have shown to be quite high in total carbon, frequently well over 4 per cent.

We are continuing the investigation with all the means at our command to determine what the remaining unknown condition may be, but it is our firm conviction that there is far more to be learned on this subject than is contained in this paper. We think that we have proved that high carbon by itself makes an iron weak and poor, but low or moderate carbon by itself is not enough. It is my hope that we shall eventually discover what causes good iron to be good, as well as what causes poor iron to be poor; and since we have been able, even with our present relatively slight degree of enlightenment, to improve the quality of our iron, and to apply different grades to the service for which they were best adapted, so we have confidence that we shall eventually succeed in producing the grades and qualities we desire, approximately at will.

This paper would not be complete without acknowledging the collaboration of G. A. Reinhardt, now instructor in metallurgy at Harvard University, who spent the summer on this research, and L. Selmi, Chief Chemist of the Ashland plant, who is responsible for all the analyses and a large number of the photomicrographs. All the detailed work of this kind has been done by these two gentlemen, without whose cordial co-operation and assistance the paper could scarcely have been written.

C. T. Dawkins, who was formerly employed by us on this investigation, was the first to make the suggestion that there was some connection between the eutectic carbon-content and the formation of spotted iron, but so far as known to me he did not claim that the spot was itself a segregation of the eutectic. The various collateral proofs of this fact adduced in the paper were developed subsequent to his departure.

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Discussion.

Prof. Henry M. Howe, New York, N. Y. (communication to the Secretary*):—Let us hope that this extremely interesting and suggestive paper may help to impress on our cast-iron metallurgists that a firm grasp of the carbon-iron diagram is necessary to a mastery of the subject. This diagram is part of our language, and we have henceforth not only to speak but to think in terms of it.

If I understand Mr. Johnson's major premise aright, I suggest a different approach to the problem. That major premise seems to be that the badness of the spotted and near-spotted irons is due to certain conditions, such as chemical composition and casting-temperature, and is to be cured by first studying rationally not so much what conditions cause badness as why they cause it, and then by removing the cause thus understood.

But the badness which counts is the badness not of the pig but of the castings made by remelting that pig; compared with this, the badness of the pig is purely academic. And the conditions of his class, composition, casting-temperature, etc., which cause the badness of the pig are not the conditions which determine the goodness or badness of the finished casting, because the casting-temperature of the remelted iron in the foundry is wholly irrelated to the casting-temperature of the pig, and because the composition of the remelted iron in the foundry differs from that of the pig in a degree which varies with the practice of the various foundries, silicon and manganese being removed, carbon and sulphur lowered or raised.

Is not the direct way to trace back the badness of the finished castings to their composition and casting-temperature; to learn what changes in composition necessarily occur when the pig is remelted under fixed and controlled good foundry conditions; and then, and only then, to give the pig a composition which, after being thus changed in remelting, will result in the composition good for castings?

The properties of cast-iron, except in so far as they are affected by the phosphorus-content, are controlled chiefly by the content of combined carbon or cementite, and of graphite, and

* Received Oct. 30, 1912.

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by its degree of hypo- or hyper-eutecticness, or, if I may coin the word, by its "eutecticity." These are the controlling causes of its properties. The manganese-, silicon-, and sulphur-contents are of importance chiefly as they affect these controlling causes. Therefore the intelligent study of this problem requires primarily a knowledge of these controlling causes; and, though the knowledge of the manganese-, silicon-, and sulphur-contents is a welcome help, we must recognize clearly that it is only a help, and that the first thing is to learn how much combined and graphitic carbon respectively are present, and the eutecticity. I believe that Mr. Johnson recognizes this, but has been prevented by analytical difficulties from giving as full light on these points as he would like.

Mr. Johnson seems to be right in the causes which he assigns to cleavage, hyper-eutecticness, and, to a less degree, eutecticness, or, in other words, primary cementite, and, to a less degree, eutectic cementite, which, except in decidedly hypo-eutectic irons, may form broad layers, the faces of which form broad cleavages. Though the primary cementite of hyper-eutectic irons, except those rich in manganese, graphitizes usually, if not always, yet the broad flakes of graphite which it yields may themselves cause broad cleavage. The eutectic cementite may itself form very broad flakes, except in decidedly hypo-eutectic irons, and these broad flakes, even if graphitized, may yield broad cleavages.

The natural protection against cleavage would seem to be marked hypo-eutecticness. This implies that large dendritic masses of primary austenite, which itself is habitually free from marked cleavage, deposit before the eutectic solidifies, with the consequence that when the eutectic does later solidify, it finds itself confined within narrow pools between the dendrites of this austenite, and its cementite can form no mass broader than these microscopic pools.

In considering eutecticity we must remember that silicon lowers the carbon-content which represents the eutectic ratio. If we infer from Wüst and Petersen that the rate of this shifting is from 4.30 per cent. of carbon for silicon zero, to 3.40 per cent. of carbon with silicon at 3 per cent., we might construct, as a formula to represent the eutectic carbon-content, $C = 4.30 - 0.3 \times Si$, in which C and Si represent the percentage of carbon

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and silicon respectively. Without some such adjustment we fall into gross errors.

It is of course hard to get a clear idea of the nature of the "spot" from verbal descriptions however graphic, and Mr. Johnson's leaves me in doubt as to whether this spot represents strong local cleavage parallel to the broken face of the pig, or a region with less graphite than the neighborhood, or both. He refers to it as a segregate. Of course he may have information which proves this, but I do not find such information in his paper. For instance, one infers from his language on p. 325 that the spot in the iron of Fig. 2 contains about 0.5 per cent. less carbon than the outside of the pig, but the data in Fig. 8 do not agree with this, giving the spot slightly more carbon than the average of the eight places tested. The spot in his hyper-eutectic iron, Fig, 7, indeed has about 0.30 per cent. less carbon than the average of the other places tested. It is true that the segregate in a hyper-eutectic iron would have less carbon than the rest; but the spot in Fig. 8 hardly seems to be a segregate, because the iron is almost exactly eutectic according to the formula given above, and a eutectic iron should not segregate. With its 0.35 per cent. of silicon the eutectic ratio should be $4.3 - 0.3 \times 0.35 = 4.195$ carbon, which is certainly very close to the average carbon-content, 4.186, with a difference of only 0.009 per cent. Therefore one wonders whether his generalization, that the spot may occur in hypo- as well as in hyper-eutectic irons, may be based on a failure to adjust for the silicon-content, and whether he has actually detected the spot in irons truly hypo-eutectic.

If the spot is simply a cleavage phenomenon, it may conceivably represent a place in the plane of rupture where the cleavage locally happens to be parallel with that plane, so that at other points in the length of the pig the spot would occur in other parts of the fracture. If this is not its nature, then the position of the spot should be nearly constant in a series of fractures at different points in the length of the pig. This point could be tested easily.

If it is a place relatively free from graphite, that is to say, if its occurrence represents a pressure, at the time of the solidification of the center, strong enough to arrest graphitization there, then one can understand its occurrence, especially in

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hyper-eutectic irons. Why should this arise in hyper- though not in hypo-eutectic irons? If I may hazard a guess it is as follows. In hypo-eutectic irons no graphitization is possible till the temperature falls to the eutectic freezing-point, and it occurs chiefly at this temperature, being retarded extremely rapidly with further fall of temperature. Hence in such irons as well as in strictly eutectic ones the graphitization is restricted to a very narrow range of temperature close below the eutectic freezing-point. In a hyper-eutectic iron graphitization starts in with the very incipiency of the solidification of primary cementite, and continues as far below the eutectic freezing-point as in the case of hypo-eutectic irons. In brief, the range of temperature through which graphitization occurs is greater in hyper- than in hypo-eutectic irons.

Graphitization implies rapid swelling, and consequent pressure. After the outer crust has solidified and determined the outer dimensions of the pig, solidification sweeps in concentric layers from the skin to the last freezing-line which we may call the axis, and with solidification goes a wave of graphitization and consequent expansion. In a hypo-eutectic iron, graphitizing in only a very narrow range of temperature, the band which at any given instant is graphitizing and setting up pressure is narrow, only the band which is then at and immediately below the eutectic freezing-point. In the hyper-eutectic iron this region is wider; there is a wider pressure-generating band sweeping from the skin to the axis, the band which at any given instant is between the temperature of the earliest precipitation of primary cementite and the temperature below the eutectic freezing-point at which graphitization becomes negligibly slow. It may be that this wider band is more effective in setting up central pressure than the narrower band of the hypo-eutectic iron. The pressure caused by the latter would seem more readily accommodated by the yielding of the crust than the cumulative pressure caused by the simultaneous expansion occurring over a wider band.

Or it may simply be that the great extent of graphitization in the hyper-eutectic iron is the direct cause of raising the pressure so high that it arrests graphitization in the axial region.

A word as to the reason why irons with more chill than their silicon-content implies are the best, and those with less chill the

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worst. Does this not represent simply the superiority of hypo- to hyper-eutectic irons? Chill means failure to graphitize in the part chilled. Abnormally great chill means restricted graphitization, deficient chill means excessive graphitization. Then the good irons are those with abnormally low graphitization, the bad irons those with abnormally high graphitization. But the degree of graphitization increases rapidly as the carbon-content rises past the eutectic ratio, for reasons explained in my paper on Ruff's Carbon-Iron Diagram, chiefly that graphitization increases rapidly as the temperature rises from the eutectic freezing-point. The graphitization of hypo-eutectic iron cannot begin till the temperature sinks to the eutectic freezing-point, whereas that of hyper-eutectic iron begins at the higher temperature at which the first of the primary cementite solidifies. The resultant graphite formed at this higher temperature acts as a nucleus to hasten the graphitization of the eutectic cementite when this solidifies on reaching the eutectic freezing-point. Hence the hyper-eutectic iron has abnormally little chill, and the strongly hypo-eutectic iron has abnormally great chill. As between different hypo-eutectic irons those decidedly hypo-eutectic graphitize less than those but slightly so, because of their greater undercooling, or in other words because of the lower temperature at which the solidification of the eutectic actually occurs. Hence they chill more deeply. Hence the abnormally deep chilling irons are exceptionally good because unusually hypo-eutectic.

The breaking up of the eutectic which Mr. Johnson refers to silicon I understand to mean graphitization of the eutectic cementite, which silicon certainly induces.

We would accede at once to his inference that even the complete graphitization of cementite would not efface the orientation which that cementite has caused, indeed would not affect it, were it not for the consideration that the resultant graphite forms idiomorphic curved flakes, instead of pseudomorphs after cementite, and it is not clear beforehand that these idiomorphic flakes need reproduce the orientation of the cementite plates which they replace. This is a point to be determined by actual observation.

Dr. R. Moldenke, Watchung, N. J. (communication to the

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Secretary *):—Mr. Johnson's paper is a highly interesting one, as forming a bridge, so to speak, between the old "mysterious" school of foundry- and furnace-men, and the newer trend of thought in which the underlying facts are looked for and mystery is brushed aside. The paper shows very ably, though probably not intentionally, what a lot of mystery there still exists when it comes to charcoal-iron opinions among foundrymen. But it is to be feared that the solution of the problem does not altogether lie in the directions Mr. Johnson's investigations have taken.

Let me say here that the able and exhaustive microscopical studies will stand by themselves as a splendid contribution to our knowledge, as also the deductions as applied to the interior chill question. But for the foundry-man, after all, the "proof of the pudding is the eating," and the car-wheel man mentioned who cast his private test-bar and broke it—provided his melting method was perfect and did not vary—did exactly what the most accomplished scientist can only do when asked to select the best material for the purpose intended.

Mr. Johnson's sixth conclusion, to my mind, explains his investigator's spirit best, when he feels that "there is another condition, element, substance, or what you will, that makes for strength and for high chill with a given silicon-content," etc. Which means that he feels that he has not yet struck the solution of the problem. So do I, and hence I beg to point out some matters which I feel will help Mr. Johnson in his investigations, just as they have proved to my own satisfaction why there is such a difference between the strength of pig-irons of "identical composition."

In the first place, when the statement is made that because two charcoal-irons of identical composition show such widely different service results, therefore there is something mysterious about it; this leaves the question, Was the chemical composition identical? In other words, does chemical analysis, as carried out to-day in even the best laboratories, show everything in the iron? My answer is that it does not. The very first application of acids may wipe out every trace of iron oxide that may be present in the sample. You have only to

* Received Oct. 30, 1912.

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turn to steel-practice to realize the vast difference in quality between a Bessemer blow cast into an ingot before the manganese addition, and a piece of cast-steel of "identical" composition, so-called, made in the open-hearth.

In all of Mr. Johnson's experiment descriptions, I see no mention of an investigation on the degree of possible oxidation existing in the several varieties of charcoal-iron. And yet a determination for oxygen, if carried out to a refinement which to my mind the importance of the subject warrants, might show that the "identical compositions" were mythical.

In my very wide experience as a foundry-man and metallurgist, I have made up my mind that nearly all the ills that we have to contend with in castings of "proper" chemical composition which either never get into service because of pin-holes, shrinkage fortunately discovered in time, etc.; or fail after they get there through interior strains, unexpected weakness, or lack of uniformity for the same heat and mixture; are due to improper melting-methods and consequent partial oxidation of the metal. I am happy to say that on the melting end, the recent investigations by the U. S. Bureau of Mines have completely borne out my contention, and I am still happier to say that the Research Laboratories of the English government are at work on the slag-inclusion, oxidation, and other problems of that class at the present moment. The importance of the subject is beginning to be realized.

Had the blast-furnace fraternity been at work on their side of the problem before this, we would not as foundry-men hear so much about the mysteries of ore varieties, coke, and other things, as affecting the resulting pig-irons made. If the hard facts were not there, as Mr. Johnson rightly remarks, to the trained metallurgist it would seem laughable to speak of "hematite," or "magnetite" made pig-iron. All molten metal of a given composition should, if cast at the same temperature and under the same conditions, give the same results. Note again "of a given composition" mentioned, for it is this that I question. Light on that subject will explain a good deal that puzzles us on the microstructure end also. It is common experience in the foundry that when burnt grate-bars have formed part of the mixture, your iron in the ladle has no "life." Iron with no "life," which means that it sets so quickly that cast-

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ings are easily lost through the freezing of the gates, is iron with an oxide of iron dissolved in it, as I have frequently proven chemically, and its freezing-point has been raised materially as a consequence. Moreover, it takes but a few hundredths of a per cent. of oxygen to do this. My opinion is that a whole line of metallographic phenomena are affected by the presence of oxidation in the samples. To revert: the burnt grate-bar metal above mentioned has an excellent chemical composition though being next to worthless as a casting. This would indicate that charcoal-irons by no means have a monopoly of the mysterious.

Now another point. Mr. Johnson does not mention that in the changing of his blast-furnace lines to get better results, this was done as the result of oxygen-determinations in the interior gases. I would remind him that tests of this kind were made in Germany which indicated the presence of free oxygen for quite a distance beyond the end of the tuyere within the furnace. In the zone of reduction, there is formed the iron sponge which subsequently melts and, depending upon the burden, comes down and out as pig-iron of one kind or another. What is to prevent a slight re-oxidation of this sponge, which we know to be most exceedingly sensitive to oxygen? My opinion is that it is the degree of this re-oxidation which makes the difference in strength and service-qualities between coke-iron and hot-, warm-, and cold-blast charcoal-irons of the "same" (*sic*) composition. I imagine that a heavily-pressed coke blast-furnace is pretty good at re-oxidizing the iron sponge, a thing which does not bother the steel-maker so much (for he uses manganese) as it does the hapless foundry-man. The comparatively lower temperature of the molten metal in the cupola allows of no subsequent correction even if it could be afforded.

My suggestion to Mr. Johnson would therefore be to look into this point, and note what his furnaces are doing in the line of producing an atmosphere absolutely free from oxygen within. It is practically impossible to get this freedom from uncombined oxygen in coke-operating cupolas; it may be so and probably is in the Southern blast-furnace, where they melt quietly, produce very small tonnages, and allow you to put your hand in the tuyeres—and incidentally get fine prices because their stock is worth it.

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A further point in Mr. Johnson's paper is the regret with which I see grading-cards mentioned. These ought by this time to be obsolete, as iron is nearly all bought by analysis—such as it is. The depth of chill gotten by one foundry in making chilled rolls or car-wheels is not the same as that gotten by another one for the same composition of iron. Indeed, the variation in temperature of casting makes a difference. And still more does the effect of oxidizing a heat slightly in the air- or open-hearth furnace by holding it too long, or slow melting, affect the chill, and melters have often wondered why at times their chill test-blocks showed so wide a difference from the actual rolls cast. Roll-makers at least, if not the car-wheel men, now know so much about their irons that no grade-card would seem desirable.

I can finally only commend Mr. Johnson's able presentation of the metallographic problems, leaving this to better posted men to discuss. If he will stick to his conclusion No. 6, and hunt for the element that he thinks missing (which I think oxygen), and which I sincerely hope he will find, the whole iron world will be the better off for his investigations.

Bradley Stoughton, New York, N. Y.:—Dr. Howe's written discussion speaks of one or two things I would like to refer to: He makes the point that he does not believe Mr. Johnson is correct in assigning these properties to pig-iron, because he says that, when the pig-iron is melted in the cupola or air-furnace, it will lose any of the qualities it has because of its hyper-eutectic composition. In the first place, I want to take issue on that very point. I think all furnace-men will agree that a bad iron will preserve its badness right through the cupola, and right through the air-furnace. If you have a particular pile of iron that gives bad castings, it will give them every time, no matter how often you melt, and the scrap, when it goes back, will again give bad castings. And not only that, but certain grades of bad iron, when they are refined and made into steel, will give bad steel. I remember very well, when I was metallurgist of a very large steel-plant, we ran into that difficulty time and time again. We found a condition which resulted in bad iron in the blast-furnace; we did not know why, but we knew that a certain condition in the blast-furnace

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made bad iron. We found that the second-class rails jumped from 2 per cent. to 15, or sometimes 18 per cent. I am informed that at present, in a very large steel-plant, the name of which I am not permitted to mention, there are certain grades of pig-iron made which will not give good steel in the open-hearth furnace.

Now, I have had certain grades of bad iron submitted to me which seemed to be perfectly correct as far as analysis was concerned, and yet would not give good castings, and until I read this paper of Mr. Johnson's I did not know why that iron was bad.

With reference to pressure forming the white spot, of which Mr. Johnson has spoken; that was discovered by Herbert E. Field, who gave the first explanation of this spot being due to pressure. Mr. Johnson, pointing out how that is impossible in general, has shown how the combination of pressure and the formation of the eutectic makes that theory perfectly reasonable and believable.

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Measurements and Relations of Hardness and Depth of Carbonization in Case-Hardened Steel.

BY MARK A. AMMON, CLEVELAND, OHIO.

(Cleveland Meeting, October, 1912.)

THE two most widely used methods of measuring hardness are the Brinell and the scleroscope. In the Brinell method a hardened steel ball is pressed into the steel under a definite load and the area of the resulting depression is measured. The load in kg. per sq. mm. of the area of the depression is taken as a measure of the hardness. Within certain limits the hardness number so determined is independent of the load or the size of the ball. In the scleroscope method a small weight carrying a diamond point is dropped through a glass tube upon the steel and the amount of rebound is taken as proportional to the hardness.

In both methods the scale of hardness is an arbitrary one; however, they agree in a comparative way, that is, a steel showing 50 per cent. increase in hardness by the Brinell test will show approximately 50 per cent. increase in hardness by the scleroscope test. Yet upon consideration of the two methods it appears that, upon a steel which has been carbonized, this should not be the case; for the scleroscope method gives almost exclusively the surface-hardness, and the condition of the material below the surface has but a slight bearing upon the hardness number. The results given by the Brinell method should, however, be the exact opposite; for the depth of penetration of a ball pressed into the steel would be dependent to a large extent upon the condition of the material below the surface. This is particularly obvious when it is considered that the size of the ball ordinarily used in the Brinell method is 1 cm. (about $\frac{3}{8}$ in.) in diameter under a load of 3,000 kg., while the diamond point of the scleroscope is in the neighborhood of 0.01 in. across, fitted in a hammer of about 0.5 oz. weight and dropping from a height of 10 inches.

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It was thought that an investigation into the hardness of a case-hardened steel by means of a combination of these two methods might lead to the development of a process whereby the depth of carbonization could be determined within a reasonable degree of commercial accuracy.

It was seen at once that the ordinary size of steel ball used in the Brinell method would not do, since the size of the depression would be so great that a finished piece would be more or less injured by it. A special attachment was therefore made for the machine, which accommodated a ball 0.25 in. in diameter, and this was used in combination with a weight of 5,000 lb. Instead of measuring the diameter of the resulting depression, the depth of penetration was measured to 0.0001 in.; and by ordinary geometrical methods the following formula was determined to give standard Brinell readings:

$$B = \frac{44{,}700}{D}$$

B being the standard Brinell reading, and D the depth of penetration in ten-thousandths of an inch.

Duplicate investigations were made upon two different steels, designated as No. 2 and No. 8, and analyzing as follows:

	No. 2. Per Cent.	No. 8. Per Cent.
Carbon	0.233	0.188
Phosphorus	0.011	0.008
Sulphur	0.019	0.020
Manganese	0.53	0.53
Nickel	...	3.30

Each of these steels was carbonized at two different temperatures, 1,650° and 1,750° F. At successive periods, the first of 0.5 hr., the second 1 hr., the third 1.5 hr., and then each hour, one piece of each grade was taken from the furnace and immediately quenched in water, while two others were allowed to cool in the carbonizing-box. The two latter were then reheated and quenched, one receiving a double and the other a single quench. The piece quenched from the box originally, received afterwards a second low heat.

The test-pieces were steel disks 1 in. thick, the two flat surfaces being ground smooth. The diameter of the disks was 1⅛ in.

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for grade No. 8, and 2 in. for No. 2. Three of each kind were packed in a cast-iron pot 8 in. in diameter and 12 in. high, making six disks in each pot. They were packed radially and at equal distances from the center. One of each grade had a wire attached to a hole drilled in it, so that it could be pulled out and quenched as soon as the box was drawn from the furnace, the other four being left in to cool with the box.

The time was divided into 10 periods as shown in the tables. Therefore 10 pots each containing the six disks as described above were placed in a very large furnace, giving throughout the heating-chamber a uniform temperature, maintained at 1,650° F. In a second furnace, exactly similar and loaded in the same way, the temperature was maintained at 1,750° F.

After each operation the disks were carefully smoothed with fine emery-cloth; three Brinell and 10 scleroscope readings were taken on each disk, and these were averaged.

In order to prevent confusion of the disks, the following scheme of marking was employed: The figure 2 or 8 was first stamped on them to distinguish the grade; this was followed by the letter H or L to distinguish the disks carbonized at a high (1,750) from a low (1,650) temperature; this was then followed by the letter B, S, or D to distinguish those quenched from the box B, those receiving a single quench S, and those receiving a double quench D after cooling in the box. After these was placed a number from 1 to 10 to represent the order in which the disks were taken from the furnace. For example, 8LS6 meant steel No. 8, carbonized at 1,650°, afterwards receiving a single quench, and the sixth disk taken from the furnace in this series.

One disk of each grade was taken from each box after cooling and a small piece was sawed out of the edge, polished, etched, and the depth of carbonization measured under the microscope, first measuring the depth of carbon greater than the eutectic, then the eutectic, and then that less than the eutectic. This measuring was done by means of a micrometer eye-piece and a mechanical stage, with a magnification of 600 diameters.

The following tabulated data give all of these results, which are reproduced in graphic form in the diagrams.

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2LB.

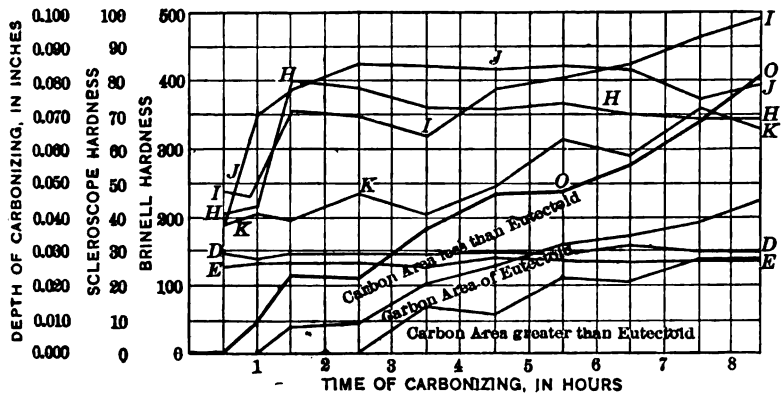
Steel No. 2. Carbonized at 1,650°. 1st Quench from Furnace at 1,650°; 2d Quench at 1,400°.

No.	Original.		1st Quench.		2d Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	136	.....	235	41	186	36
2	128	.....	229	43	205	70
3	128	.....	358	80	196	78
4	135	.....	344	78	235	85
5	131	.....	319	72	226	66
6	131	.....	388	72	246	84
7	128	.....	407	74	315	85
8	135	.....	425	71	292	84
9	139	.....	470	70	372	75
10	122	.....	496	68	331	80

2LS.

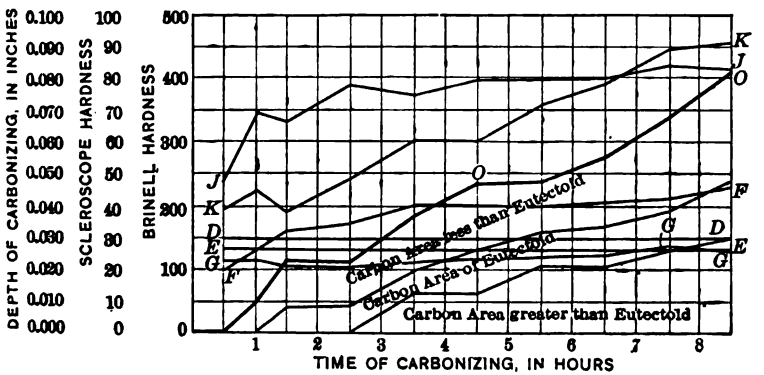
Steel No. 2. Carbonized at 1,650°; Quenched at 1,400°.

No.	Original.		After Carb.		1st Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	126	29	112	20	194	48
2	131	27	112	25	224	70
3	131	29	106	32	190	67
4	131	29	109	35	242	78
5	128	29	114	40	302	75
6	141	30	119	40	302	80
7	137	29	122	40	358	80
8	132	32	124	41	388	80
9	135	30	140	42	447	85
10	129	30	128	45	456	83



2LB. Steel No. 2 Carbonized at 1,650°.

1st quench from furnace at 1,650° (Scler. hardness = *H*; Brin. hardness = *I*). 2d quench at 1,400° (Scler. hardness = *J*; Brin. hardness = *K*).



2LS. Steel No. 2 Carbonized at 1,650°.

(Scler. hardness = *F*; Brin. hardness = *G*). Quenched at 1,400° (Scler. hardness = *J*; Brin. hardness = *K*).

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2LD.

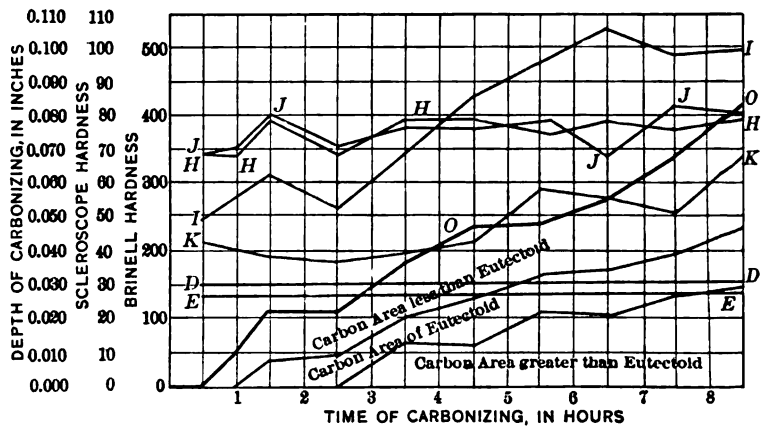
Steel No. 2. Carbonized at 1,650°. 1st Quench at 1,650°; 2d Quench at 1,400°.

No.	Original.		1st Quench.		2d Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	128	.....	248	68	211	68
2	130	.....	271	68	201	70
3	135	.....	304	78	188	78
4	126	.....	260	68	183	70
5	135	.....	344	78	196	76
6	136	.....	426	78	213	76
7	135	.....	470	74	375	78
8	136	.....	520	78	268	62
9	131	.....	486	75	251	62
10	136	.....	496	78	331	80

2HB.

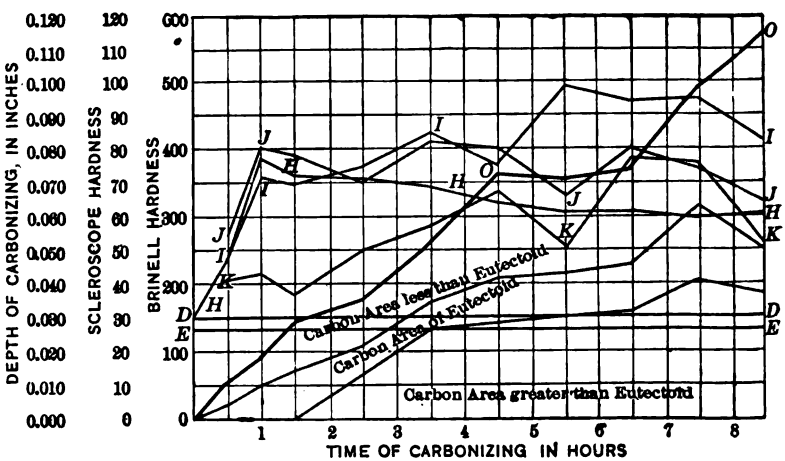
Steel No. 2. Carbonized at 1,750°. 1st Quench from Furnace at 1,750°; 2d Quench at 1,400°.

No.	Original.		1st Quench.		2d Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	128	.....	232	46	205	53
2	130	.....	358	78	213	80
3	128	.....	344	72	185	78
4	130	.....	372	71	248	70
5	128	.....	425	69	285	82
6	130	.....	372	64	338	88
7	131	.....	496	61	256	66
8	131	.....	470	61	388	80
9	137	.....	480	60	379	75
10	135	.....	414	61	263	63



2LD. Steel No. 2 Carbonized at 1,650°.

1st quench at 1,650° (Scler. hardness = *H*; Brin. hardness = *I*). 2d quench at 1,400° (Scler. hardness = *J*; Brin. hardness = *K*).



2HB. Steel No. 2 Carbonized at 1,750°.

1st quench from furnace at 1,750° (Scler. hardness = *H*; Brin. hardness = *I*). 2d quench at 1,400° (Scler. hardness = *J*; Brin. hardness = *K*).

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2HS.

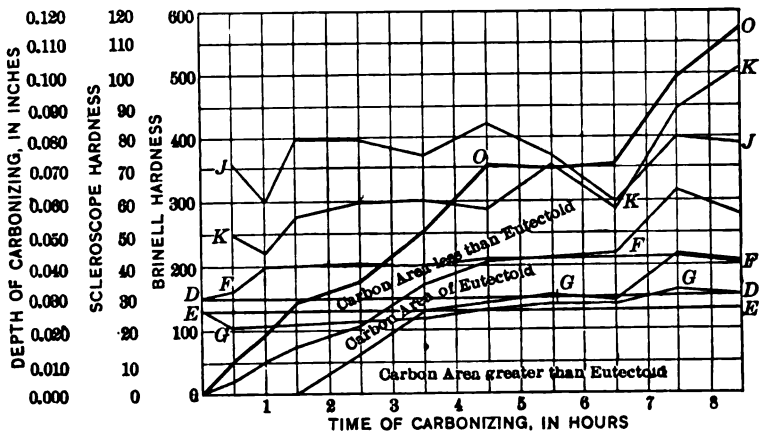
Steel No. 2. Carbonized at 1,750°. 1st Quench at 1,400°

No.	Original.		After Carb.		1st Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	133	30	106	32	248	72
2	135	31	106	40	218	60
3	128	32	110	40	279	80
4	129	30	110	41	302	80
5	131	30	117	40	302	75
6	134	30	131	41	288	85
7	131	31	140	42	358	75
8	128	29	142	42	288	60
9	138	30	160	42	447	80
10	138	29	149	40	508	78

2HD.

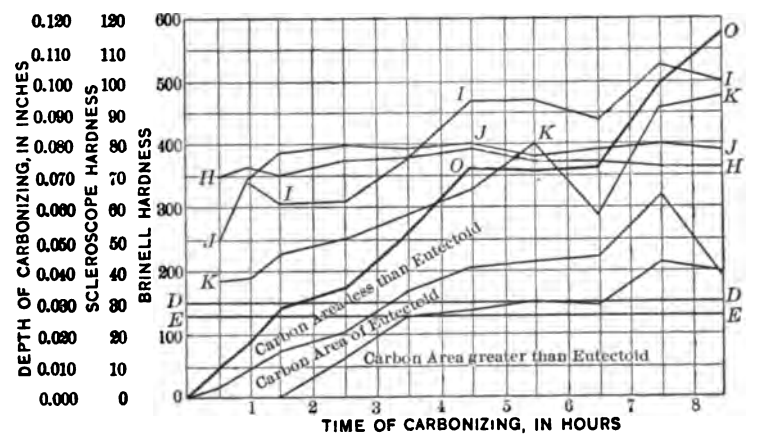
Steel No. 2. Carbonized at 1,750°. 1st Quench at 1,650°; 2d Quench at 1,400°.

No.	Original.		1st Quench.		2d Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	135	28	425	70	186	48
2	133	24	341	72	190	70
3	136	27	308	70	224	78
4	135	30	310	75	344	80
5	128	32	372	75	288	78
6	129	35	470	78	324	80
7	131	35	470	74	399	76
8	133	37	438	74	280	78
9	129	30	526	72	457	80
10	130	29	496	72	470	78



2HS. Steel No. 2 Carbonized at 1,750°.

Scler. hardness = *F*; Brin. hardness = *G*. 1st quench at 1,400° (Scler. hardness = *J*; Brin. hardness = *K*).



2HD. Steel No. 2 Carbonized at 1,750°.

1st quench at 1,650° (Scler. hardness = *H*; Brin. hardness = *I*). 2d quench at 1,400° (Scler. hardness = *J*; Brin. hardness = *K*).

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8LB.

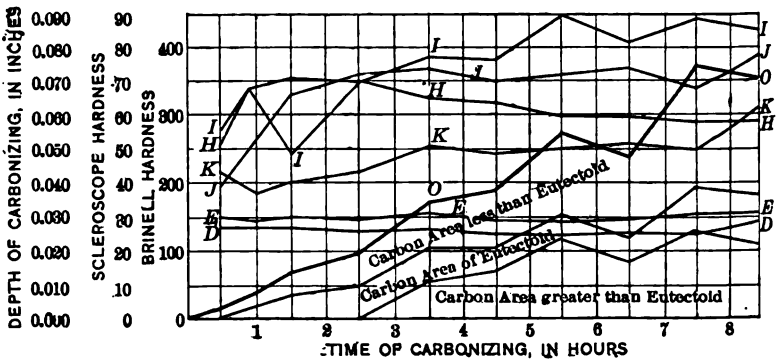
Steel No. 8. Carbonized at 1,650°, 1st Quench from Furnace at 1,650°; 2d Quench at 1,325°.

No.	Original.		1st Quench.		2d Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	151	27	285	53	213	40
2	144	27	338	68	199	53
3	149	28	246	71	203	65
4	149	26	352	70	218	72
5	154	25	388	65	255	74
6	144	25	382	64	212	70
7	144	25	447	60	251	72
8	147	25	407	60	260	74
9	154	25	447	58	248	68
10	154	26	425	58	312	78

8LS.

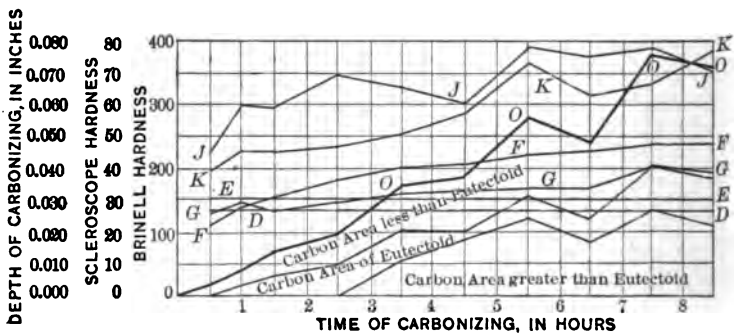
Steel No. 8. Carbonized at 1,650°. 1st Quench at 1,325°.

No.	Original.		After Carb.		1st Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	153	.....	131	23	198	45
2	154	.....	144	28	226	60
3	154	.....	131	31	224	60
4	155	.....	149	36	235	70
5	151	.....	161	41	251	65
6	149	.....	161	41	288	60
7	149	.....	168	44	366	78
8	149	.....	167	45	313	75
9	149	.....	198	47	324	77
10	150	.....	190	47	379	70



8LB. STEEL NO. 8 CARBONIZED AT 1,650°.

1st quench from furnace at 1,650° (Scler. hardness = *H*; Brin. hardness = *I*). 2d quench at 1,325° (Scler. hardness = *J*; Brin. hardness = *K*).



8LS. STEEL NO. 8 CARBONIZED AT 1,650°.

Scler. hardness = *F*; Brin. hardness = *G*. Quenched at 1,325°. (Scler. hardness = *J*; Brin. hardness = *K*).

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8LD.

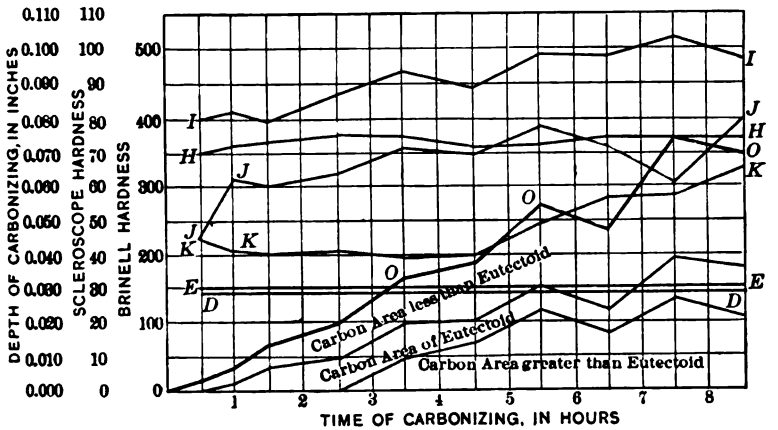
teel No. 8. Carbonized at 1,650°. 1st Quench at 1,575°; 2d Quench at 1,325°.

No.	Original.		1st Quench.		2d Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	151	25	396	70	221	46
2	151	28	407	72	201	62
3	154	28	398	73	201	60
4	155	28	438	75	206	64
5	160	28	470	75	196	72
6	151	28	447	72	201	70
7	153	27	496	72	256	78
8	151	28	496	75	288	70
9	153	29	526	75	292	62
10	153	29	496	70	330	80

8HB.

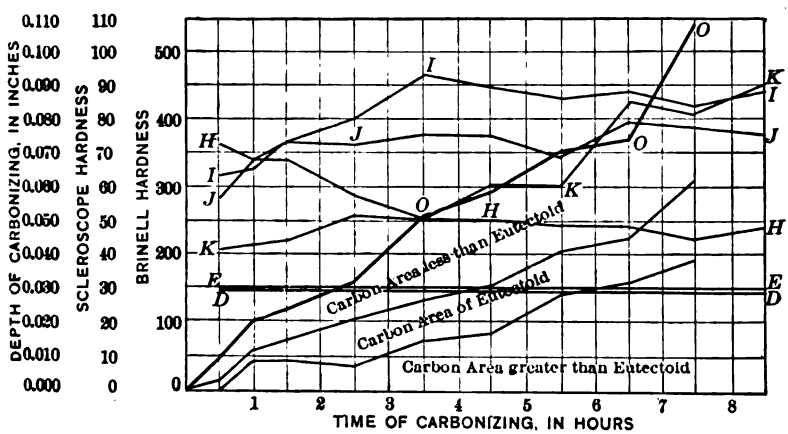
Steel No. 8. Carbonized at 1,750°. 1st Quench from Furnace at 1,750°: 2d Quench at 1,325°.

No.	Original.		1st Quench.		2d Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	154	32	315	72	208	56
2	147	30	326	68	211	68
3	142	29	366	68	218	73
4	139	30	398	57	255	72
5	147	28	466	50	251	75
6	142	30	447	50	298	75
7	139	30	425	48	300	68
8	139	30	438	48	425	78
9	144	32	418	44	407	77
10	147	34	438	47	447	75



8LD. Steel No. 8 Carbonized at 1,650°.

1st quench at 1,575° (Scler. hardness = *H*; Brin. hardness = *I*). 2d quench at 1,325° (Scler. hardness = *J*; Brin. hardness = *K*).



8HB. Steel No. 8 Carbonized at 1,750°.

1st quench from furnace at 1,750° (Scler. hardness = *H*; Brin. hardness = *I*) 2d quench at 1,325° (Scler. hardness = *J*; Brin. hardness = *K*).

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8HS.

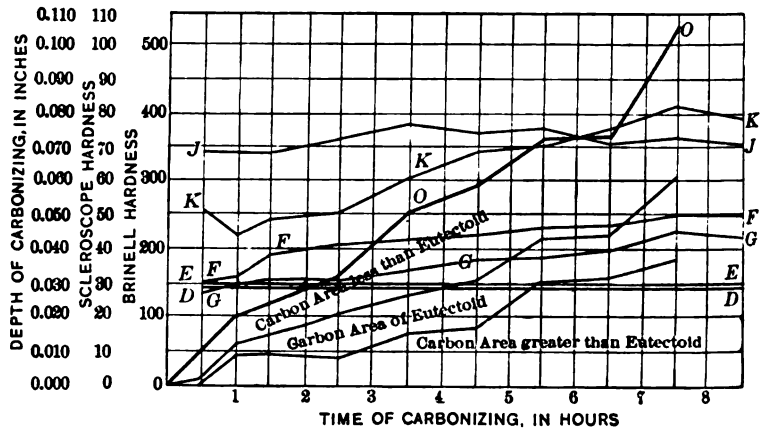
Steel No. 8. Carbonized at 1,750°. 1st Quench at 1,325°.

No.	Original.		After Carb.		1st Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	154	29	135	30	256	68
2	145	29	148	32	218	68
3	151	28	154	38	242	68
4	162	31	154	41	251	72
5	151	30	168	42	302	76
6	154	31	182	44	239	73
7	157	31	186	46	350	75
8	151	29	194	46	372	70
9	155	30	224	50	407	72
10	149	30	212	47	389	70

8HD.

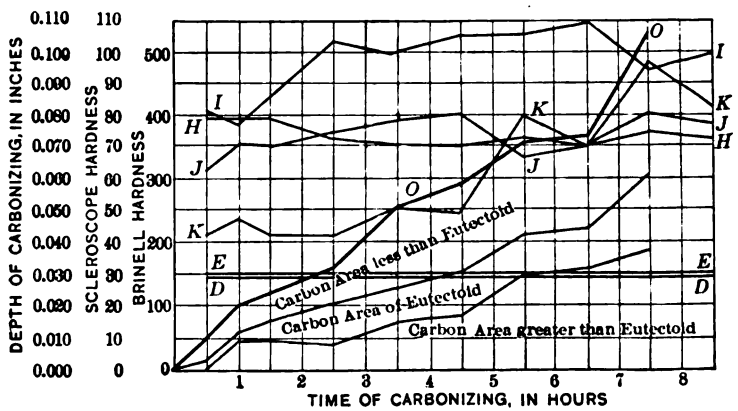
Steel No. 8. Carbonized at 1,750°. 1st Quench at 1,575°; 2d Quench at 1,325°.

No.	Original.		1st Quench.		2d Quench.	
	Brin.	Scle.	Brin.	Scle.	Brin.	Scle.
1	146	.....	407	78	208	62
2	149	.....	382	78	233	70
3	153	.....	425	78	208	70
4	155	.....	508	72	208	74
5	151	.....	496	70	251	78
6	154	.....	526	70	242	80
7	158	.....	526	72	396	66
8	148	.....	546	70	349	70
9	151	.....	470	75	485	80
10	149	.....	496	72	407	76



8HS. Steel No. 8 Carbonized at 1,750°.

(Scler. hardness = *F*; Brin. hardness = *G*). Quenched at 1,325° (Scler. hardness = *J*; Brin. hardness = *K*).



8HD. Steel No. 8 Carbonized at 1,750°.

1st quench at 1,575° (Scler. hardness = *H*; Brin. hardness = *I*). 2d quench at 1,325° (Scler. hardness = *J*; Brin. hardness = *K*).

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Discussion of Results.

Allowing for a few discrepancies, the results show a consistent uniformity in demonstrating certain facts. I will discuss these in detail:

Original Scleroscope and Brinell Hardness.

These hardnesses are designated by the letters *D* and *E* respectively on all the diagrams. On 2LB and 8LB are plotted the individual hardnesses of the original steel before carbonizing. It will be noted that these lines are practically horizontal, for which reason the original hardness on the other diagrams was taken as the average of all the pieces, and therefore the lines are drawn horizontally.

Depth of Carbonization.

This is plotted in the three areas as previously described and printed on the drawings. The line *O* denotes the maximum depth of carbonization. Under like conditions of carbonization, the rate of penetration in the carbon-steel was 1.07 times the rate in the nickel-steel at the low (1,650°) temperature. At the high temperature (1,750°) this rate for the carbon was 1.15 times that for nickel.

These carbonization-depth curves show the characteristic flattening out and subsequent rise after the initial period of carbonization.

The average rate of penetration of the carbon-steel at the high temperature was 1.48 times the rate at the low temperature, while for the nickel-steel the figure was 1.41, showing the tremendous influence of temperature.

Hardness after Carbonizing and before Quenching.

See 2HS, 2LS, 8HS and 8LS. The most apparent fact which we note here, is the higher scleroscope reading after carbonizing compared with the reading before carbonizing, while the corresponding Brinell readings are but slightly higher in grade No. 8 and even lower in grade No. 2. This may be readily explained. The scleroscope method, being influenced largely by the surface-conditions, here measures the increased hardness due to the higher carbon. It will be noted that there is a sharp rise in this curve at the beginning until the surface-carbonization

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is greater than 0.90 per cent. and from here on the increase in hardness is more gradual. The Brinell hardness, being more a function of the core than of the surface, is actually made less for the lower depths of carbon, due to the soft annealing produced by the slow cooling in the carbonizing-boxes. However, as the depth of carbonization increases, this Brinell reading is more and more influenced by it and less by the conditions of the core. A careful consideration of the facts thus brought out will show that the No. 8 steel was better annealed originally than No. 2.

Hardness after Quenching from Carbonizing-Box.

See 2HB, 2LB, 8HB, 8LB. The scleroscope readings show a gradual decrease in hardness with increased depth in carbon, after the first hour of carbonization. For both grades the steel carbonized at the higher temperature shows the greater decrease in hardness with increasing depth of case; and No. 8, for the same temperature of carbonization, shows a lower hardness than No. 2. The explanation is, that, with increasing depths of carbonization the percentage of carbon in the surface becomes higher; in quenching from this high temperature the austenite-forming tendencies increase with increasing depth; and the higher the percentage of austenite, the lower the hardness. For the same steel, the higher temperature of carbonization produces more cementite and consequently austenite; and likewise, at the same temperature, the carbon-steel produces more cementite than the nickel-steel. This explanation can be verified by comparing the scleroscope hardnesses with those produced when the steel is quenched at a lower temperature and the excess of cementite is allowed to remain out of solution.

Let us now note the Brinell hardnesses corresponding to the above scleroscope hardnesses. These show some slight irregularities, undoubtedly due to differences in cooling-rate caused by mechanical difficulties of quenching from the large carbonizing-boxes. However, the curves show an increase in hardness from 4 to 6 hr. and then the hardness becomes practically stationary. This is in direct contrast to the scleroscope hardness. The probabilities are that these Brinell readings would have begun to decrease if longer length of time had been used. This is actually shown in 2HB and 8HB. The explanation of

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this Brinell hardness is, that the influence of the core decreases while the influence of the case increases as the depth of the carbon becomes greater; therefore, as soon as the depth becomes great enough to cause the core-influence to be negligible, the hardness ceases to rise and then decreases, for the same reason as explained in the discussion of the scleroscope readings.

Hardness after First Quench on Double-Quenched Specimens.

See 2LD, 2HD, 8LD, 8HD. The scleroscope readings after this quench are practically horizontal. The gradual reduction in hardness shown in the higher quench from the box is absent. It will be noted that this hardness is considerably higher than that obtained upon quenching direct from the box, by reason of the weaker austenite-forming tendencies. The Brinell hardness shows the same increase as for those quenched from the box, and also the tendency to begin dropping towards the end. The actual Brinell hardness is also about the same as that obtained when quenching from the box.

Hardness after Second Quench.

See 2LD, 2HD, 8LD, 8HD. We will now discuss those cooled in the box and then receiving a double quench. Let us first consider what we should expect from a theoretical standpoint. For surfaces of less than 0.90 per cent. of carbon, we should expect a lower hardness than when quenched from a higher temperature, by reason of the fact that an excess of ferrite is now present. In proportion as a higher percentage of carbon than 0.90 is obtained, we should expect a harder surface, due to the fact that we have allowed an excess of cementite to appear. A critical study of the curves indicates that this is the tendency, although the difference is slight. Now, considering the Brinell hardness after the second quench, we here note a very remarkable dropping off in hardness. The hardness at the lower depths of carbonization is approaching the original hardness, and with increasing depth of carbon it shows a decided increase. This is due to the fact that at the low temperature of quenching the core is no longer in the martensitic condition and therefore does not resist the penetration of the Brinell ball; but as the depth of carbon increases the core exerts less and less influ-

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ence, and the case exerts more, with a consequent rise in the hardness. It will be noted that the hardness does not reach the hardness of the first quench except for the higher heats of carbonization. It is interesting to note that we have here all the data necessary for a determination of the depth of carbonization at which the condition of the core ceases to have any influence upon the Brinell hardness. This is approximately at the point where the Brinell reading on the second quench is the same as that on the first quench. This is about 0.11 in. deep.

The hardness on the second quench of the steel quenched from the box is similar to that of the second quench of the tests just considered.

Hardness after a Single Low Heat.

See 2LS, 2HS, 8LS, 8HS. The scleroscope line here exhibits the same characteristics as previously explained for low heats, except for some irregularities caused by the fact that the cementite here occurs in large masses rather than in disseminated grains. The Brinell line is also similar to that obtained on the second quench of the double heat, with the exception that on the low heats of carbonization it rises faster and to a greater figure than on the double quench. This is undoubtedly an influence of the core, although it is hard to explain unless it is caused by the difference in the condition of dissemination of the ferrite and martensite in the two cases.

The knowledge gained by the above research-work has formed the basis of a system of testing carbonized parts for depth of carbonization and efficiency of the heat-treatment.

Discussion.

J. E. Johnson, Jr., Ashland, Wis.:—Mr. Ammon's paper is of great value and it is not with any desire to reflect on it that I wish to make some comments on the use of the scleroscope.

We obtained one on trial some months ago and made very extensive tests with it, but without getting very satisfactory results. The readings on good car-wheels and bad ones were just about the same, but the factor in its use most likely to produce erroneous results is a variation in the thickness of the specimen being tested.

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In order to find out something on this subject we cut a thin disk, about $\frac{1}{16}$ in. thick, off a brass rod; then one 0.25 in. thick, and another about 1 in. thick, put the surfaces of all in about the same condition and tested them with the scleroscope. The readings varied widely; the resilience of the anvil evidently entered to a very high degree on thin pieces. By putting a soft cushion between such a thin specimen and the anvil, and forcing it up against the tube, the softness of the cushion cut off the resilience of the anvil and the results were more like those of the thick specimens.

After a thickness of 1 in. was passed, the specimen seemed to have mass enough so that the resilience of the anvil no longer played an important part, and above that thickness results on different materials would be comparable, even though the thicknesses differed. But in order to get comparable results on thin specimens the thickness of those being tested should be the same. I should like to ask how the variations introduced in this way were overcome, so as to get strictly comparable results.

MR. AMMON:—I have not had much experience in that, but the scleroscope is simply an indication of the surface hardening. You cut them on an anvil of any ordinary firm basis, and you could not detect any practical difference, but to take a piece $\frac{3}{16}$ or $\frac{1}{8}$ in. thick, I have not had any experience on those lines at all. But the scleroscope is simply to show the elasticity of the surface.

BRADLEY STOUGHTON, New York, N. Y.:—I presume the reason Mr. Ammon did not find any differences was because the material was harder or more resilient than the anvil. This subject has come up very prominently before the American Society for Testing Materials. Now, that society has a committee which studies the question of hardness, and it was admitted by Mr. Shore himself, the inventor of the scleroscope, that thin pieces did not give good results. I presume, however, that if the thin pieces are harder than the anvil, the anvil would not affect them; but if they are softer, in a case where the pieces are $\frac{3}{16}$ of an in. thick, the effect of the anvil would be shown.

I want to say that there is a very great deal of information

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on the question of carbonizing and case-hardening in general, and hardening tests, and I suppose 80 per cent. of it has never been made public. I think that the Peerless Motor Company, to which these men belong, ought to be thanked by us for allowing the results that have been given here to be published, because we sincerely hope it will form a nucleus around which we can gather a great deal more valuable information on this subject. Case-hardening is no doubt going through an important transitional stage, being improved almost daily; and anything that can be done to get the many results in the shops made public, is of very great value to the industry to which this matter is so important.

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The Action of Various Commercial Carbonizing-Materials.

BY ROBERT R. ABBOTT, E. M., MET. E.,* CLEVELAND, OHIO.

(Cleveland Meeting, October, 1912.)

THE practice of carbonizing steel for the purpose of case-hardening has assumed great commercial importance within the past 10 years.

Formerly, case-hardened steel was held in more or less contempt, since it was considered a cheap substitute for tool-steel. This is no longer the case. The development of the motor-car industry, and coincidentally that of modern alloy-steels, has resulted in overcoming many difficulties formerly experienced in case-hardening. This result has been accomplished by increased knowledge regarding the analysis of the steel which would respond most readily to case-hardening, and by more careful methods of treatment of the carbonized steel.

The ordinary method of carbonizing consists in packing the steel with the carbonizing-material in cast-iron boxes and placing them in a furnace at a temperature of from 1,500° to 1,900° F. for a time sufficient to give the required depth of "case." The steel is then either quenched directly from the box, or is allowed to cool without unpacking, and finally given a single hardening-heat, or, for special work, two or three heats.

During the year 1911, more than 100,000 tons of carbonizing-material was sold in the United States, at an average price of probably $55 per ton. More than 85 per cent. of this material consisted of granulated bone.

Within the past four years, many manufactured compounds have been placed on the market, consisting largely of some form of carbon or carbonaceous material, with or without the addition of chemicals.

Steel will absorb carbon placed in contact with it at the temperature of the atmosphere. The reaction is exceedingly slow, but increases rapidly with increasing temperature. Below

* Metallurgical Engineer, The Peerless Motor Car Co.

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1,500° F., the absorption is too slow to be commercially important. Various causes combine to make it impracticable to carbonize above 1,900° F. In general, carbonizing is carried out at temperatures ranging from 1,550° to 1,750°, and higher temperatures are used only when the quality is not as important as the cost. Broadly speaking, the higher temperatures can be employed for high-grade work when proper facilities are at hand for a careful regulation of the temperature, and a knowledge is possessed of the correct subsequent heat-treatment of the carbonized product.

When steel is carbonized, the carbon does not penetrate in a gradually decreasing content from a high-carbon exterior to the uncarbonized core, but rather in a series of steps; for example, there is usually present a considerable zone of the eutectoid composition, which in the present paper has been assumed to contain 0.90 per cent. of carbon.

Many of the commercial materials behave differently in carbonizing, and even under exactly the same conditions of time and temperature different depths of penetration or different per cents. of carbon, or both, are obtained.

In order to produce good results in case-hardening, a uniform material must be used, and the treatment of the steel subsequent to carbonizing must be suited to the nature of the case produced.

The present investigation was undertaken to compare most of the important commercial carbonizers as to cost of carbonizing, rapidity of carbonizing, and the nature of the resultant carbon-zones.

The method of procedure usually followed in investigating carbonizing-compounds consists in carbonizing small bars for different lengths of time, and then turning off successive layers, usually 0.010 in. deep, and analyzing the turnings for carbon. This method is not entirely satisfactory, as two bars showing practically the same carbon-content might be carbonized in entirely different ways, so that the same subsequent heat-treatment would produce decidedly different results.

Method of Investigation.

The method used in the investigation is the result of more than four years' experience in studying carbonizing and car-

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bonizing-compounds. Twenty-five materials were investigated, designated by the letters *A* to *Y*. For obvious reasons, the names of the compounds are withheld.

Materials *C*, *D*, *I*, and *U* are pure bone obtained from different firms. *U* and *I* consist of bone in which the glue has been partly or wholly extracted. *A* and *K* are bone mixed with other carbonaceous material. *E* consists of partly-roasted nuts; *W* consists largely of husks and kernels of seeds or beans. *X* and *Y* consist of a charred carbonaceous material heavily charged with chemicals.

The remainder of the compounds consist of carbonaceous material in the form of graphite, coal, coke, and charcoal, most of which are charged with various chemicals.

(1) Eight bars of steel, all from the same heat, were selected after check-analysis showed them to be of uniform composition. The analysis of this material was: C, 0.205; P, 0.015; S, 0.025; Mn, 0.53; Si, 0.04 per cent.

These bars, 0.75 in. in diameter, were cut in 0.75-in. lengths. The pieces, each weighed carefully on an analytical balance, were placed in small white cast-iron pots, surrounded by carbonizing-material, as shown in Fig. 1. The cover of the pot was luted with fire-clay. The pots were new, and the same pot was used throughout with the material used in it at the start. Identical tests were carried out on each material at three different temperatures, 1,600°, 1,750°, and 1,900° F., respectively.

Ten pots of each material were packed for each temperature, and the 250 pots were placed on the hearth of a gas-fired furnace, having a hearth of such a width and length that the entire bank of pots was 2 ft. from either end and 8 in. from either side. The pots were arranged in 10 rows of 25 to a row, each row contained one pot of each material, and these separate rows were placed upon a strip of sheet-iron, so that the entire row could be drawn out at the end of each hour. Each strip containing 25 pots was first placed on a large plate of sheet-iron, and the furnace was brought up to heat and kept 25° F. above the chosen temperature for 2 hr. before the heat was placed in it. The furnace was provided with a door at each end, and at the proper time the large plate containing the 250 pots was drawn into the furnace. The furnace-temperature was then maintained constant for 10 hr. Two pyrometer-couples, used for

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measuring the temperature, were connected to a recording-instrument, which gave a record of each couple every 40 sec. At the end of an hour, the first row of pots was drawn from the furnace. The operation of drawing this strip of sheet-iron upon which the pots were placed occupied about 30 sec. At the end of 2 hr., the next row was drawn, etc. The mass of the brick-work in the furnace used was sufficiently great to prevent sudden changes of temperature. After the pots were cool, the pieces of steel were taken from them, carefully washed and reweighed. They were then sawn in two, polished, etched, and examined under the microscope.

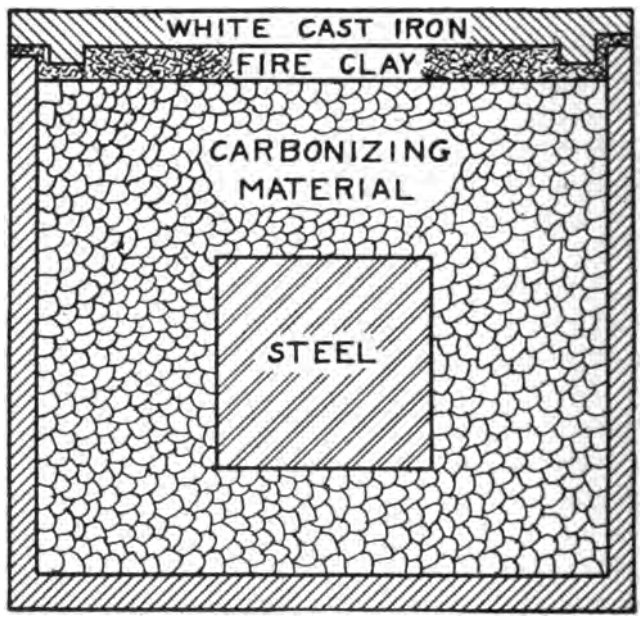

FIG. 1.—A CARBONIZING POT. CROSS-SECTION.

By the use of a micrometer eye-piece, the carbonized surface was measured. First, the depth of the zone of carbon containing more than the eutectic ratio (which was considered as 0.90 per cent.) was measured, next the zone containing the eutectic ratio. The remainder of the carbonization was then divided into two zones, the line of division being estimated as the point which contained 0.50 per cent. of carbon. The percentage of carbon in the extreme outer surface was also estimated. The

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results of these measurements, weights, and estimations are given in Tables VII. to XXXI.

Materials *X* and *Y* caused a decided pitting of the surface of the steel, and in many cases it was impossible to weigh the pieces accurately.

The results regarding the depths of carbonization and increase in weight, shown graphically in the diagrams of Tables VII. to XXXI., are designated by the letter corresponding to the material, and by the temperature of carbonization; thus *A*–1,600 represents graphically the depths of the various zones of carbon produced by material *A* at a temperature of 1,600° for different lengths of time from 1 to 10 hr. The abscissa, or base-line, represents the surface of the steel. The ordinates erected to this at each hour represent the depth of carbonization. The hypereutectic zone of carbon is distinguished by single cross-hatching; the eutectic zone, by double cross-hatching. The two zones of hypoeutectic carbon are not cross-hatched, nor are the zones of decarbonization. (See *C*–1,750.) The broken line represents the increase in weight.

Repeated Use of Material.

An attempt was made to determine the action of the different materials under repeated use. For this purpose, the material used for 9 and 10 hr. at 1,750° was saved and repacked in two more pots, two being repacked because the shrinkage would finally require more material than the single pot could hold. This second heat was then run for 10 hr. at 1,750°, repacked, and run for two additional heats of 10 hr. This test, of which the results are given in Tables VII. to XXXI., was not entirely satisfactory, however, and is now being repeated on a much larger scale.

The repeated use of material is open to serious objection when uniform results are desired. The ordinary method of mixing part old and part new material is objectionable for the reason that in most places where a great variety of carbonizing is done, the old material is not uniform, since it has been used at different temperatures and for different lengths of time.

The efficiencies considered in various ways, under one use only, and for the three temperatures, of the different materials, are tabulated in Tables I. to VI.

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TABLE I.—*Relative Cost of Depositing the Same Weight of Carbon with Different Materials at Three Different Temperatures.*

1,600° F.		1,750° F.		1,900° F.	
Material.	Cost.	Material.	Cost.	Material.	Cost.
V	$1.00	*V*	$1.00	*V*	$1.00
R	1.30	*R*	1.28	*Q*	1.21
M	1.99	*Q*	1.54	*R*	1.38
Q	3.00	*M*	1.73	*M*	1.81
E	4.00	*L*	3.26	*S*	3.43
H	4.54	*S*	3.97	*L*	3.51
J	4.55	*W*	4.24	*W*	3.89
B	4.71	*E*	4.30	*E*	3.89
A	4.85	*H*	4.31	*H*	4.47
I	4.92	*J*	4.60	*P*	4.52
U	5.18	*Y*	4.62	*J*	4.60
P	5.30	*A*	4.70	*A*	5.22
D	5.30	*P*	4.90	*B*	5.22
C	5.84	*B*	5.30	*N*	5.40
F	7.11	*I*	5.40	*O*	5.43
N	7.18	*U*	5.83	*Y*	6.60
S	8.00	*O*	6.37	*I*	6.64
O	8.10	*D*	6.44	*X*	6.78
K	11.71	*N*	6.54	*U*	7.07
.....	.....	*F*	7.06	*F*	7.20
.....	.....	*C*	7.33	*D*	7.65
.....	.....	*K*	10.60	*C*	9.43
.....	.....	*T*	34.10	*K*	9.65
.....	.....	.....	.....	*T*	32.40

These efficiencies were calculated in the following manner: The total increase in weight of the 10 pieces for each temperature was divided by the total number of hours of carbonizing, namely, 55. This gave the average increase in weight for that temperature per hour. (This is a fair average, for in most plants material will be carbonized for different purposes for lengths of time varying from 1 to 10 hr.) This average increase in weight was then used as a basis of comparison in three different ways, which will be explained. In the same way, the total depth of carbonization for each material under each temperature was divided by 55, thus giving an average depth of penetration per hour.

From the commercial standpoint, the material which will deposit the most carbon in the steel at a given cost is the most efficient. Table I. has been calculated upon this basis. Material *V*, being the most efficient (for all three temperatures), was given the value $1. The table is then interpreted thus: by using material *R* (at 1,600°), it will cost $1.30 to deposit the

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TABLE II.—*Relative Cost of Carbonizing to a Given Depth with Different Materials at Three Different Temperatures.*

1,600° F.		1,750° F.		1,900° F.	
Material.	Cost.	Material.	Cost.	Material.	Cost.
V	$1.00	*V*	$1.00	*Q*	$1.00
R	1.41	*Q*	1.07	*V*	1.06
M	2.04	*R*	1.48	*R*	1.54
Q	2.43	*M*	2.07	*M*	2.16
H	4.36	*S*	2.97	*S*	3.03
P	4.57	*L*	3.12	*L*	3.60
E	4.65	*W*	3.76	*W*	3.95
I	4.85	*P*	4.61	*P*	4.86
B	4.95	*H*	4.90	*H*	5.07
A	5.02	*J*	5.18	*E*	5.60
J	5.15	*E*	5.35	*J*	5.72
U	5.19	*X*	5.37	*O*	5.86
X	5.20	*Y*	5.52	*X*	6.12
Y	5.20	*A*	5.52	*B*	6.17
D	5.55	*I*	5.57	*Y*	6.19
S	5.73	*U*	5.63	*A*	6.25
C	6.15	*B*	5.76	*I*	6.26
O	6.33	*O*	6.48	*U*	6.27
N	7.88	*D*	6.54	*D*	7.61
F	8.88	*C*	7.00	*N*	7.80
K	10.10	*N*	7.77	*C*	8.61
.....	.....	*F*	9.05	*K*	9.88
.....	.....	*K*	9.87	*F*	9.93
.....	.....	*T*	34.75	*T*	44.75

same weight of carbon as could be deposited by material *V* at a cost of $1. Likewise, material *K* will cost $11.71 under the same conditions.

Table II. is determined in the same manner, with the exception that instead of being calculated upon a basis of efficiency in depositing a given weight of carbon, a given depth of penetration was used.

Table III. gives the relative efficiency of the same weight of different materials in depositing a given weight of carbon; that is, in a determination of the average weight of carbon deposited by the same weight of different materials, that material is the most efficient which will deposit the most carbon. Material *R* was found to be the most efficient in this respect, and was given the value of 100 per cent. *M* was second, with the efficiency of 81.6 per cent. at 1,600° F. As a simple illustration, this can be interpreted as follows: 1 lb. of material *M* will deposit 81.6 per cent. as much carbon as 1 lb. of material *R*.

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TABLE III.—*Relative Efficiency of the Same Weight of Different Materials in Depositing a Given Weight of Carbon at Three Different Temperatures.*

1,600° F.		1,750° F.		1,900° F.	
Material.	Efficiency.	Material.	Efficiency.	Material.	Efficiency.
	Per Cent.		Per Cent.		Per Cent.
R	100.0	*R*	100.0	*R*	100.0
M	81.6	*M*	92.3	*M*	95.8
V	81.2	*V*	79.8	*V*	86.5
J	66.4	*Y*	69.1	*J*	69.8
E	57.0	*L*	68.6	*L*	69.0
B	44.8	*J*	64.5	*E*	62.2
F	41.2	*E*	52.0	*N*	57.5
N	40.6	*N*	44.0	*Q*	57.2
A	38.8	*B*	43.2	*X*	53.7
D	31.8	*Q*	41.3	*Y*	52.4
I	31.4	*F*	40.7	*F*	43.2
U	31.3	*A*	39.4	*B*	43.1
P	30.6	*T*	33.7	*S*	40.5
C	28.9	*P*	32.6	*A*	38.4
H	26.3	*S*	32.4	*T*	38.3
Q	21.6	*W*	30.2	*P*	38.3
O	20.0	*H*	29.6	*W*	35.6
K	16.6	*I*	28.0	*O*	31.8
S	16.2	*U*	27.4	*H*	31.0
G	12.2	*D*	25.8	*I*	24.8
.....	.....	*O*	25.1	*U*	24.5
.....	.....	*C*	22.6	*D*	23.5
.....	.....	*K*	18.2	*K*	21.5
.....	.....	*G*	13.9	*C*	19.1
.....	.....	.....	.....	*G*	16.4

Table IV. is calculated the same way as Table III., with the exception that its efficiency is based upon the depth of carbonization given by the same weight of different materials.

Table V. is based upon the relative efficiencies of the same volume of different materials in depositing a given weight of carbon. In this case, material *F* is the most efficient at 1,600° and 1,750° F.; while material *N* is the most efficient at 1,900° F. This table of efficiencies does not take into account the specific gravity of the different materials, and, therefore, other things being equal, the heaviest material will appear as the most efficient.

Table VI. is similar to Table V., except that the depth of carbonization is the basis and not the weight.

Discussion of Results.

A critical discussion of the results of efficiencies given in Tables I. to VI. could be extended indefinitely. We will, there-

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TABLE IV.—*Relative Efficiency of the Same Weight of Different Materials in Carbonizing to a Given Depth at Three Different Temperatures.*

1,600° F.		1,750° F.		1,900° F.	
Material.	Efficiency.	Material.	Efficiency.	Material.	Efficiency.
	Per Cent.		Per Cent.		Per Cent.
R	100.0	*R*	100.0	*R*	100.0
V	88.0	*V*	92.5	*V*	91.0
M	86.3	*M*	89.9	*M*	89.5
X	71.1	*L*	83.3	*Q*	77.2
Y	67.8	*X*	72.5	*L*	75.0
J	63.5	*Q*	69.1	*X*	66.2
E	52.8	*Y*	67.2	*J*	62.8
B	46.3	*J*	66.4	*Y*	62.3
A	40.7	*S*	50.0	*S*	50.9
N	40.2	*E*	48.5	*E*	48.3
P	38.4	*N*	42.8	*N*	44.4
F	35.3	*B*	41.7	*B*	40.6
I	34.5	*P*	40.3	*P*	39.8
U	33.9	*W*	39.3	*W*	39.1
D	32.9	*A*	39.0	*A*	35.9
H	32.3	*T*	38.4	*F*	35.0
C	29.8	*F*	36.9	*O*	32.9
Q	28.9	*U*	32.9	*T*	31.1
O	27.8	*I*	31.6	*U*	30.8
S	24.5	*H*	30.3	*H*	30.5
K	20.9	*D*	29.4	*I*	29.3
G	11.0	*O*	28.6	*D*	26.4
.....	.....	*C*	27.6	*K*	23.5
.....	.....	*K*	22.6	*C*	23.3
.....	.....	*G*	19.3	*G*	22.8

fore, consider but a few examples to illustrate the method of making practical use of the data.

The use to which case-hardened steel is to be put should be the deciding factor in selecting the material with which it is to be carbonized, and also to a less extent the temperature of carbonization should be determined from similar considerations.

Assume that a plant was making brake-shoes which required tremendous resistance to wear and but slight resistance to shock. The pressure upon the brake-shoe would likewise be distributed over a considerable area and would not be concentrated in a point or on a line, with the resulting tendency to bend the carbonized surface in a manner similar to the bending of a beam by a load at the center. Under this condition, the main requirements should be a surface containing the maximum amount of carbon possible, since under the proper subse-

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TABLE V.—*Relative Efficiency of the Same Volume of Different Materials in Depositing a Given Weight of Carbon at Three Different Temperatures.*

1,600° F.		1,750° F.		1,900° F.	
Material.	Efficiency.	Material.	Efficiency.	Material.	Efficiency.
	Per Cent.		Per Cent.		Per Cent.
F	100.0	*F*	100.0	*N*	100.0
E	99.8	*E*	92.9	*E*	96.0
R	89.0	*Y*	92.5	*F*	92.3
B	88.0	*R*	90.1	*J*	81.0
J	87.7	*N*	87.5	*T*	79.8
C	86.2	*M*	87.0	*R*	78.5
A	84.5	*A*	86.9	*M*	77.7
V	82.4	*J*	86.4	*V*	77.1
D	82.3	*B*	86.2	*B*	74.5
U	81.9	*V*	81.8	*A*	73.6
N	79.9	*T*	80.7	*O*	69.8
I	78.0	*U*	72.4	*W*	66.4
M	75.2	*I*	70.7	*H*	65.3
H	63.0	*C*	68.3	*P*	65.2
P	59.2	*D*	67.2	*Y*	61.0
K	50.5	*H*	65.3	*X*	59.5
O	49.8	*W*	64.8	*K*	57.6
G	30.9	*L*	64.1	*U*	56.2
S	22.4	*P*	64.0	*L*	56.0
Q	19.3	*O*	63.3	*I*	54.2
.....	.....	*K*	55.7	*D*	53.3
.....	.....	*S*	45.0	*C*	50.0
.....	.....	*Q*	37.3	*S*	49.3
.....	.....	*G*	35.8	*Q*	44.8
.....	.....	.....	.....	*G*	36.9

quent treatment this would give the best wearing-surface. Consider, for an example, the relative efficiencies of materials *F* and *K* for this purpose. Assume that a temperature of 1,750° F. was to be used. Both materials would give approximately the same depths of carbon. However, material *K*, excepting for 10 hr., gives a surface carbonization which is not in excess of 0.90 per cent., while *F* gives a surface carbonization of about 1.10 per cent. Material *F* would evidently be far better than *K*. If the surface was to be ground after hardening, most of the high carbon would be removed from *F*, leaving the surface-carbon considerable lower. However, this same grinding would remove most of the 0.90 per cent. of carbon from the steel carbonized with material *K*, and in this case also material *F* would be the better.

Assume that a gear was to be case-hardened. We might

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TABLE VI.—*Relative Efficiency of the Same Volume of Different Materials in Carbonizing to a Given Depth at Three Different Temperatures.*

1,600° F.		1,750° F.		1,900° F.	
Material.	Efficiency.	Material.	Efficiency.	Material.	Efficiency.
	Per Cent.		Per Cent.		Per Cent.
E	100.0	*V*	100.0	*V*	100.0
B	98.1	*X*	97.0	*R*	96.4
C	96.8	*T*	96.6	*N*	94.6
Y	96.7	*F*	95.5	*F*	92.0
X	96.7	*R*	95.0	*E*	91.6
V	96.3	*Y*	94.5	*X*	90.5
R	96.2	*J*	92.5	*J*	89.8
A	95.8	*U*	91.5	*W*	89.8
U	95.5	*A*	90.4	*M*	89.3
F	93.5	*E*	90.4	*Y*	89.1
I	92.6	*N*	89.6	*O*	88.6
D	92.0	*W*	88.6	*U*	87.1
J	90.5	*C*	88.1	*B*	86.5
M	85.9	*M*	88.1	*A*	84.6
N	85.2	*B*	87.3	*P*	83.3
H	83.6	*I*	83.6	*T*	79.6
P	80.4	*P*	82.8	*H*	79.2
O	74.6	*L*	81.5	*I*	79.0
K	68.7	*D*	80.6	*K*	77.4
S	36.6	*H*	77.2	*S*	76.2
G	30.3	*O*	75.6	*C*	76,0
Q	27.8	*S*	73.5	*L*	74.8
.....	.....	*K*	73.0	*Q*	74.4
.....	.....	*Q*	65.6	*D*	73.8
.....	.....	*G*	52.4	*G*	62.8

here consider two possible cases: First, that the work was being done by a firm which had very slight knowledge of the heat-treatment necessary to put the gear in the best condition after carbonization; and, second, that the work was being done by a firm which had the necessary knowledge. Suppose material *V* was used by both firms and at a temperature of 1,750° F. After the gear was carbonized and allowed to cool in the carbonizing-pot, the surface would consist of hypereutectic steel; that is, it would contain spines or a net-work of brittle cementite or iron carbide surrounding the cells of pearlite, which contained the eutectic ratio of carbon, *i. e.*, 0.90 per cent. The firm without the necessary heat-treatment knowledge would reheat this gear to the hardening-temperature, which they would find to be slightly less than 1,400° F. This would leave the spines or net-work of cementite unchanged, and as a result

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the surface would be traversed by this extremely brittle material, which, if the carbonization were not deep enough, might allow a dangerous deflection, due to the concentrated load which must be carried by the carbonized surface, acting in a manner similar to a beam. This would cause minute fractures in the surface, which would finally result in its being pitted and chipped. If, however, the gear was correctly heat-treated, as assumed in the second case, this cementite would be broken up and distributed throughout the surface in minute particles, which would have no embrittleing effect upon it, but would act as particles of extremely hard, wear-resisting material.

In the case above assumed, it would be better that a material similar to *K* or *P* be used in the first case, since, by the use of this material, the outer zone of brittle cementite would not be formed, and although the surface would not be as hard, due to the absence of free cementite, still the tendency towards brittleness would be eliminated.

It will be noted in Table II. that material *Q* stands near the top as regards the efficiency of cost, being the most efficient of all the materials at 1,900° F.

An examination of Table XXIII. shows that at 1,600° no 0.90 per cent. carbon is produced; therefore, steel carbonized with this material at this temperature could not be hardened sufficiently to give a good wearing-surface. This is practically true of 1,750° also. At 1,900°, however, considerable 0.90 per cent. carbon is produced, and if this high temperature was not objectionable, this material would be efficient if nothing else were considered. However, upon comparing *V*–1,900 to *Q*–1,900, it is seen that *V* will deposit a much greater amount and also a greater depth of carbon than *Q* in the same length of time, and will give also a much deeper zone of eutectic and hypereutectic carbon. It is evident, then, that, considering the cost due to the longer time necessarily required by *Q* to give the same depth of carbon, it would probably be considerably less efficient from a practical standpoint.

The question of carbonizing-material is one which usually is not given the attention which its importance demands.

A few sample tests conducted under intelligent direction will often result not only in a great saving of money, but in increased efficiency with regard to the product.

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TABLE VII.—*Material "A."*

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1,600° F.							
1	0.0225	0.90	0.0000	0.0048	0.0012	0.0024	0.0084
2	0.043	0.92	0.0048	0.0072	0.0060	0.0048	0.0228
3	0.0515	0.95	0.0048	0.0060	0.0048	0.0048	0.0204
4	0.064	0.95	0.0060	0.0060	0.0120	0.0072	0.0312
5	0.0645	0.95	0.0072	0.0120	0.0168	0.0024	0.0384
6	0.0745	0.95	0.0072	0.0060	0.0120	0.0084	0.0336
7	0.083	0.98	0.0072	0.0120	0.0108	0.0120	0.0420
8	0.0795	1.00	0.0096	0.0072	0.0168	0.0024	0.0360
9	0.088	0.95	0.0072	0.0120	0.0180	0.0084	0.0456
10	0.0845	1.00	0.0084	0.0120	0.0180	0.0096	0.0480
1,750° F.							
1	0.0355	0.92	0.0036	0.0036	0.0060	0.0048	0.0180
2	0.0585	1.00	0.0084	0.0096	0.0060	0.0072	0.0312
3	0.072	1.00	0.0072	0.0096	0.0120	0.0048	0.0336
4	0.0955	1.00	0.0156	0.0084	0.0180	0.0084	0.0504
5	0.114	1.05	0.0192	0.0156	0.0144	0.0132	0.0624
6	0.1215	1.10	0.0180	0.0132	0.0240	0.0060	0.0612
7	0.143	1.15	0.0204	0.0180	0.0192	0.0108	0.0684
8	0.1395	0.95	0.0084	0.0300	0.0300	0.0072	0.0756
9	0.148	1.25	0.0204	0.0144	0.0288	0.0060	0.0696
10	0.1475	1.00	0.0144	0.0264	0.0288	0.0180	0.0876
1,900° F.							
1	0.063	0.90	0.0000	0.0132	0.0120	0.0036	0.0288
2	0.082	0.90	0.0000	0.0144	0.0132	0.0132	0.0408
3	0.059	0.90	.....	0.0252	0.0192	0.0144	0.0588
4	0.128	0.90	.....	0.0522	0.0348	0.0156	0.0756
5	0.1485	1.05	0.0096	0.0312	0.0252	0.0216	0.0876
6	0.1495	1.05	0.0228	0.0228	0.0252	0.0096	0.0804
7	0.191	0.90	.....	0.0492	0.0444	0.0168	0.1104
8	0.2045	1.00	0.0252	0.0564	0.0336	0.0216	0.1368
9	0.223	1.10	0.0216	0.0456	0.0492	0.0192	0.1356
10	0.236	1.00	0.0268	0.0420	0.0588	0.0120	0.1392
Repeated Use at 1,750° F.							
1st 10	0.1475	1.00	0.0144	0.0264	0.0288	0.0180	0.0876
2d 10	0.1540	1.00	0.0204	0.0192	0.0360	0.0132	0.0888
3d 10	0.1130	0.95	0.0072	0.0168	0.0240	0.0192	0.0672
4th 10	0.097	0.90	.....	0.0252	0.0312	0.0132	0.0696

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TABLE VIII.—*Material "B."*

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1,600° F.							
1	0.021	0.90	.....	0.0048	0.0024	0.0024	0.0096
2	0.0395	0.90	.....	0.0060	0.0060	0.0060	0.0180
3	0.0505	0.93	0.0054	0.0048	0.0120	0.0012	0.0234
4	0.0575	0.95	0.0072	0.0072	0.0096	0.0084	0.0324
5	0.0625	0.90	.....	0.0120	0.0180	0.0048	0.0348
6	0.069	1.05	0.0084	0.0036	0.0180	0.0084	0.0384
7	0.074	0.92	0.0048	0.0120	0.0180	0.0060	0.0408
8	0.078	0.93	0.0060	0.0108	0.0180	0.0024	0.0372
9	0.0825	0.93	0.0060	0.0120	0.0180	0.0120	0.0480
10	0.087	0.93	0.0054	0.0204	0.0240	0.0018	0.0516
1,750° F.							
1	0.035	0.90	.....	0.0036	0.0108	0.0096	0.0240
2	0.0545	0.90	.....	0.0096	0.0096	0.0084	0.0276
3	0.068	0.95	0.0084	0.0096	0.0168	0.0048	0.0396
4	0.0805	0.95	0.0120	0.0072	0.0132	0.0072	0.0396
5	0.1125	1.15	0.0132	0.0168	0.0144	0.0072	0.0516
6	0.123	1.10	0.0156	0.0168	0.0192	0.0084	0.0600
7	0.135	1.00	0.0192	0.0216	0.0120	0.0072	0.0600
8	0.143	1.10	0.0192	0.0216	0.0288	0.0060	0.0756
9	0.1535	1.15	0.0228	0.0156	0.0252	0.0132	0.0768
10	0.162	1.10	0.0216	0.0240	0.0264	0.0120	0.0840
1,900° F.							
1	0.058	0.90	.....	0.0108	0.0192	0.0084	0.0384
2	0.086	1.00	0.0060	0.0156	0.0168	0.0108	0.0492
3	0.109	0.90	.....	0.0300	0.0204	0.0084	0.0588
4	0.124	0.95	0.0048	0.0336	0.0264	0.0168	0.0816
5	0.144	0.93	0.0096	0.0384	0.0228	0.0060	0.0768
6	0.155	1.00	0.0204	0.0252	0.0288	0.0228	0.0972
7	0.185	1.10	0.0108	0.0420	0.0420	0.0168	0.1116
8	0.199	1.10	0.0240	0.0300	0.0504	0.0168	0.1212
9	0.216	1.00	0.0156	0.0480	0.0432	0.0312	0.1380
10	0.228	1.00	0.0228	0.0384	0.0420	0.0372	0.1404
Repeated Use at 1,750° F.							
1st 10	0.162	1.10	0.0216	0.0240	0.0264	0.0120	0.0840
2d 10	0.120	0.95	0.0108	0.0240	0.0276	0.0192	0.0816
3d 10	0.109	0.90	.....	0.0216	0.0324	0.0120	0.0660
4th 10	0.109	0.95	0.0060	0.0204	0.0360	0.0156	0.0780

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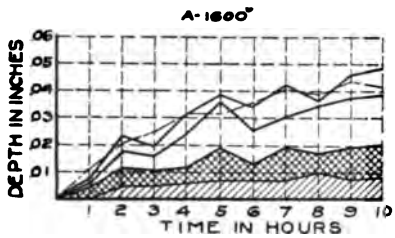

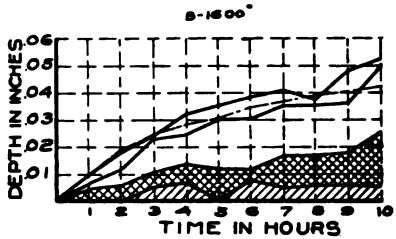

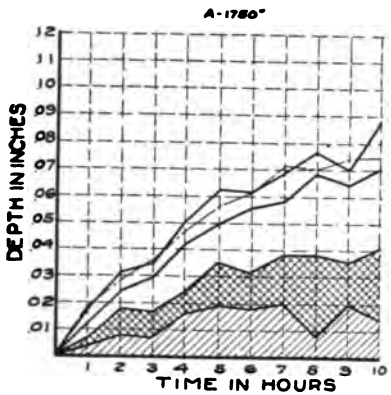

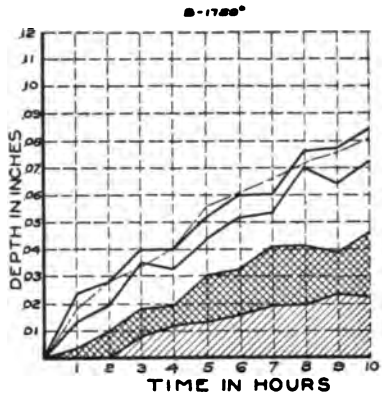

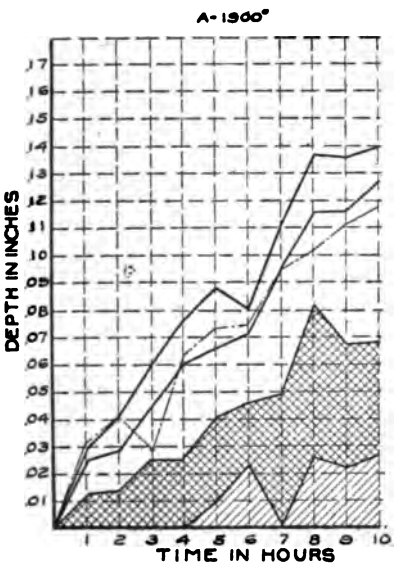

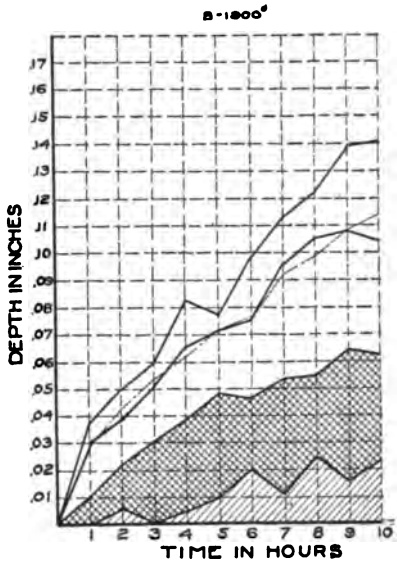

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TABLE IX.—*Material "C."*

1,600° F.

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1	0.022	0.90	.....	0.0048	0.0036	0.0012	0.0096
2	0.030	0.90	.....	0.0048	0.0060	0.0036	0.0144
3	0.0495	0.93	0.0060	0.0048	0.0120	0.0012	0.0240
4	0.058	1.00	0.0072	0.0048	0.0120	0.0048	0.0288
5	0.064	1.00	0.0060	0.0072	0.0144	0.0060	0.0336
6	0.0695	0.95	0.0060	0.0120	0.0180	0.0060	0.0420
7	0.0755	1.00	0.0072	0.0096	0.0180	0.0096	0.0444
8	0.0735	0.95	0.0060	0.0144	0.0120	0.0072	0.0396
9	0.083	1.00	0.0096	0.0048	0.0240	0.0108	0.0492
10	0.0845	0.95	0.0060	0.0144	0.0120	0.0120	0.0444

1,750° F.

1	0.0385	0.90	.....	0.0084	0.0048	0.0048	0.0180
2	0.0595	0.92	0.0072	0.0084	0.0108	0.0024	0.0288
3	0.070	0.95	0.0084	0.0108	0.0156	0.0084	0.0432
4	0.085	1.05	0.0096	0.0120	0.0180	0.0048	0.0444
5	0.1045	1.15	0.0132	0.0204	0.0180	0.0024	0.0540
6	0.115	0.92	0.0084	0.0228	0.0204	0.0180	0.0696
7	0.085	0.80	.....	.....	0.0552	0.0072	0.0624
8	0.0875	0.00	.....	0.0264	0.0324	0.0072	0.0660
9	0.113	0.80	.....	.....	0.0660	0.0192	0.0852
10	0.087	0.20	.....	.....	0.0636	0.0084	0.0720

1,900° F.

1	0.051	0.90	.....	0.0108	0.0240	0.0048	0.0396
2	0.072	0.90	.....	0.0156	0.0252	0.0144	0.0552
3	0.071	0.95	0.0084	0.0132	0.0264	0.0096	0.0576
4	0.103	0.90	.....	0.0252	0.0264	0.0252	0.0768
5	0.1175	0.90	.....	0.0228	0.0408	0.0144	0.0780
6	0.113	0.90	.....	0.0264	0.0420	0.0276	0.0960
7	0.139	1.00	0.0144	0.0276	0.0288	0.0144	0.0852
8	0.119	0.00	.....	0.0408	0.0444	0.0312	0.1164
9	0.099	0.20	.....	.....	0.0576	0.0240	0.0816
10	0.125	0.00	.....	.....	0.0852	0.0312	0.1164

Repeated use at 1,750° F.
The material would not carbonize after the first ten-hour heat.

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TABLE X.—*Material "D."*

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1,600° F.							
1	0.023	0.90	.....	0.0048	0.0024	0.0024	0.0096
2	0.0295	0.80	.....	.....	0.0096	0.0056	0.0132
3	0.056	1.00	0.0060	0.0024	0.0084	0.0024	0.0192
4	0.0525	0.95	0.0060	0.0072	0.0120	0.0024	0.0276
5	0.059	0.93	0.0072	0.0060	0.0120	0.0060	0.0312
6	0.0875	0.95	0.0060	0.0108	0.0120	0.0084	0.0372
7	0.067	0.90	.....	0.0144	0.0240	0.0060	0.0444
8	0.077	0.90	.....	0.0156	0.0180	0.0084	0.0420
9	0.061	0.70	.....	.....	0.0300	0.0132	0.0432
10	0.068	0.90	.....	0.0144	0.0240	0.0072	0.0456
1,750° F.							
1	0.036	0.90	.....	0.0060	0.0060	0.0060	0.0180
2	0.0555	0.95	0.0072	0.0060	0.0096	0.0048	0.0276
3	0.0695	1.10	0.0096	0.0108	0.0096	0.0060	0.0360
4	0.0865	1.15	0.0132	0.0120	0.0168	0.0060	0.0480
5	0.1065	1.10	0.0120	0.0156	0.0168	0.0060	0.0504
6	0.113	1.10	0.0192	0.0168	0.0240	0.0084	0.0684
7	0.1115	0.90	.....	0.0300	0.0240	0.0060	0.0600
8	0.099	0.80	.....	.....	0.0660	0.0072	0.0732
9	0.077	0.00	.....	.....	0.0540	0.0108	0.0648
10	0.078	0.00	.....	.....	0.0360	0.0144	0.0504
1,900° F.							
1	0.051	0.90	.....	0.0108	0.0144	0.0072	0.0324
2	0.073	0.90	.....	0.0120	0.0264	0.0132	0.0516
3	0.080	0.90	.....	0.0264	0.0180	0.0144	0.0588
4	0.112	0.90	.....	0.0480	0.0108	0.0156	0.0744
5	0.170	0.90	.....	0.0288	0.0408	0.0132	0.0828
6	0.125	0.95	0.0096	0.0240	0.0396	0.0252	0.0984
7	0.090	0.70	.....	.....	0.0840	0.0204	0.1044
8	0.109	0.20	.....	.....	0.0648	0.0168	0.0816
9	0.160	0.90	.....	0.00576	0.0360	0.0180	0.1116
10	0.105	0.20	.....	.....	.....	0.0840	0.0840
Repeated Use at 1,750° F.							
2d 10	0.016	0.20	.....	.....	.....	slight	.....

This material would not carbonize after this heat.

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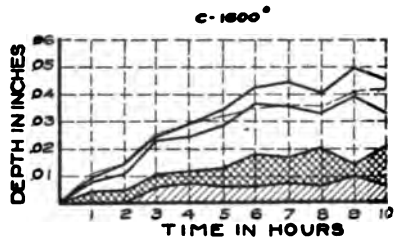

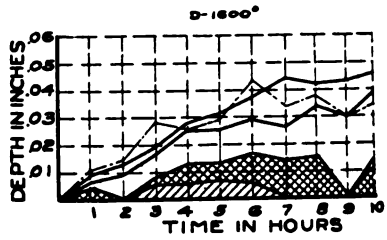

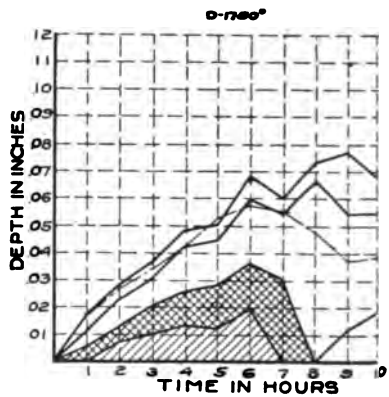

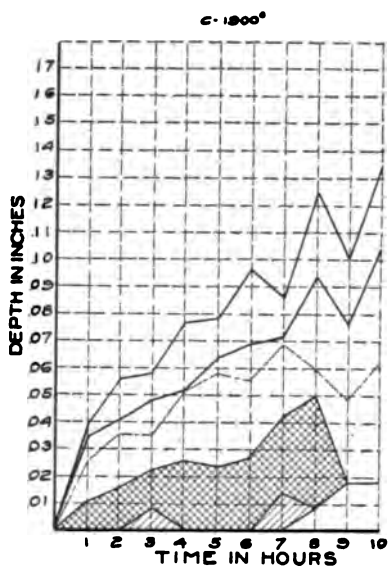

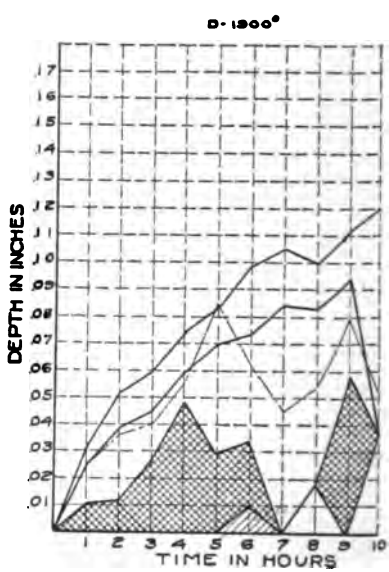

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TABLE XI.—*Material "E."*

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1,600° F.							
1	0.027	0.90	.....	0.0048	0.0048	0.0012	0.0108
2	0.039	0.90	.....	0.0060	0.0060	0.0036	0.0156
3	0.059	0.90	.....	0.0072	0.0060	0.0012	0.0144
4	0.065	0.93	0.0060	0.0060	0.0096	0.0048	0.0264
5	0.070	0.92	0.0048	0.0108	0.0120	0.0084	0.0360
6	0.082	0.95	0.0120	0.0072	0.0240	0.0072	0.0504
7	0.096	0.95	0.0072	0.0084	0.0192	0.0060	0.0408
8	0.084	1.00	0.0084	0.0108	0.0180	0.0060	0.0432
9	0.072	1.20	0.0132	0.0072	0.0240	0.0024	0.0468
10	0.110	0.90	.....	0.0192	0.0240	0.0132	0.0564
1,750° F.							
1	0.042	0.92	0.0048	0.0072	0.0060	0.0060	0.0240
2	0.072	1.05	0.0120	0.0060	0.0072	0.0060	0.0312
3	0.088	1.00	0.0120	0.0096	0.0168	0.0048	0.0432
4	0.098	1.05	0.0144	0.0168	0.0060	0.0048	0.0420
5	0.094	1.10	0.0108	0.0120	0.0108	0.0084	0.0420
6	0.138	1.15	0.0228	0.0168	0.0180	0.0072	0.0648
7	0.144	1.15	0.0216	0.0168	0.0204	0.0096	0.0684
8	0.146	1.15	0.0228	0.0192	0.0216	0.0132	0.0768
9	0.170	1.20	0.0204	0.0300	0.0228	0.0096	0.0828
10	0.158	1.20	0.0276	0.0204	0.0300	0.0048	0.0828
1,900° F.							
1	0.093	1.10	0.0132	0.0132	0.0048	0.0060	0.0372
2	0.124	1.15	0.0180	0.0156	0.0156	0.0108	0.0600
3	0.156	1.10	0.0276	0.0132	0.0240	0.0048	0.0696
4	0.173	1.15	0.0276	0.0240	0.0276	0.0084	0.0876
5	0.198	1.20	0.0300	0.0240	0.0360	0.0096	0.0996
6	0.227	1.25	0.0408	0.0216	0.0288	0.0136	0.1068
7	0.212	1.15	0.0408	0.0252	0.0336	0.0168	0.1164
8	0.211	1.20	0.0372	0.0240	0.0324	0.0072	0.1008
9	0.268	1.25	0.0396	0.0480	0.0420	0.0120	0.1416
10	0.274	1.20	0.0420	0.0444	0.0420	0.0192	0.1476
Repeated Use at 1,750°.							
1st 10	0.158	1.20	0.0276	0.0204	0.0300	0.0048	0.0828
2d 10	0.136	1.10	0.0168	0.0204	0.0216	0.0144	0.0732
3d 10	0.087	0.95	0.0144	0.0168	0.0240	0.0120	0.0672
4th 10	0.121	0.95	0.0108	0.0264	0.0180	0.0096	0.0648

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TABLE XII.—*Material "F."*

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1,600° F.							
1	0.020	0.90	.....	0.0036	0.0048	0.0024	0.0108
2	0.040	0.90	.....	0.0072	0.0060	0.0024	0.0156
3	0.050	0.90	.....	0.0060	0.0060	0.0048	0.0168
4	0.070	0.90	.....	0.0108	0.0168	0.0012	0.0288
5	0.077	1.00	0.0072	0.0024	0.0168	0.0072	0.0336
6	0.082	1.00	0.0066	0.0036	0.0144	0.0096	0.0342
7	0.088	0.92	0.0072	0.0192	0.0060	0.0048	0.0372
8	0.083	1.00	0.0084	0.0060	0.0180	0.0072	0.0396
9	0.088	1.10	0.0084	0.0144	0.0180	0.0036	0.0444
10	0.108	0.93	0.0120	0.0132	0.0240	0.0084	0.0576
1,750° F.							
1	0.040	1.00	0.0108	0.0120	0.0036	0.0024	0.0288
2	0.070	1.10	0.0084	0.0096	0.0060	0.0072	0.0312
3	0.079	1.10	0.0120	0.0108	0.0120	0.0048	0.0396
4	0.101	1.00	0.0156	0.0108	0.0156	0.0084	0.0504
5	0.101	1.20	0.0168	0.0108	0.0180	0.0084	0.0540
6	0.138	1.05	0.0216	0.0240	0.0144	0.0060	0.0660
7	0.169	1.10	0.0240	0.0180	0.0240	0.0096	0.0756
8	0.166	1.05	0.0252	0.0348	0.0096	0.0036	0.0732
9	0.180	1.05	0.0312	0.0180	0.0240	0.0084	0.0816
10	0.194	1.20	0.0264	0.0264	0.0240	0.0120	0.0888
1,900° F.							
1	0.076	1.10	0.0084	0.0108	0.0144	0.0072	0.0408
2	0.108	1.05	0.0132	0.0180	0.0204	0.0144	0.0660
3	0.140	1.15	0.0180	0.0192	0.0180	0.0168	0.0720
4	0.166	1.15	0.0240	0.0288	0.0228	0.0168	0.0924
5	0.198	1.15	0.0240	0.0300	0.0300	0.0180	0.1020
6	0.200	1.20	0.0336	0.0288	0.0240	0.0156	0.1020
7	0.212	1.15	0.0384	0.0180	0.0360	0.0108	0.1032
8	0.218	1.15	0.0396	0.0288	0.0288	0.0192	0.1164
9	0.258	1.20	0.0420	0.0420	0.0300	0.0144	0.1284
10	0.286	1.20	0.0612	0.0264	0.0420	0.0192	0.1488
Repeated Use at 1,750°.							
1st 10	0.194	1.20	0.0264	0.0264	0.0240	0.0120	0.0888
2d 10	0.167	1.10	0.0264	0.0240	0.0300	0.0060	0.0864
3d 10	0.053	1.10	0.0276	0.0180	0.0240	0.0192	0.0888
4th 10	0.063	1.10	0.0288	0.0144	0.0372	0.0132	0.0936

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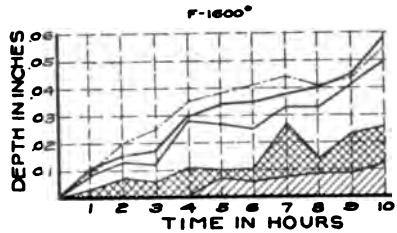

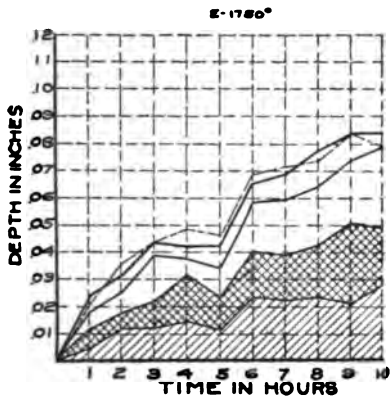

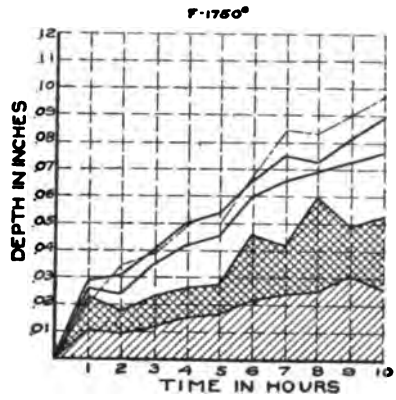

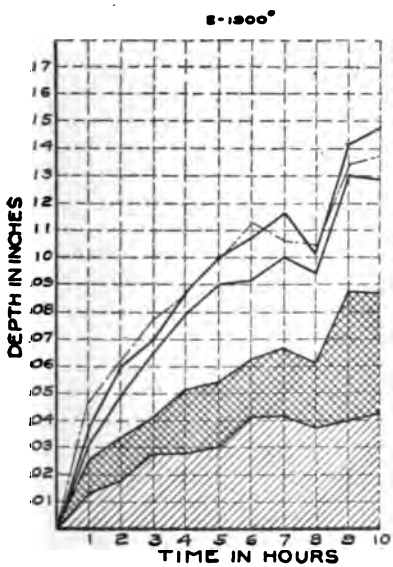

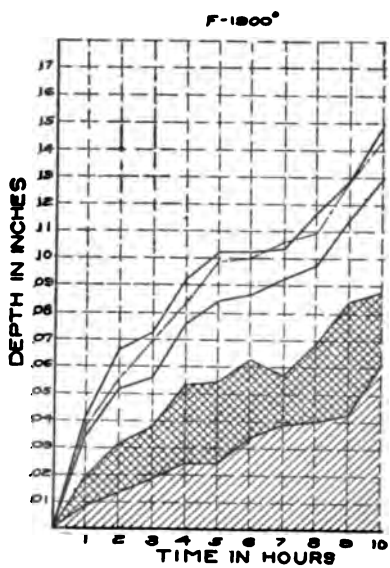

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TABLE XIII.—*Material "G."*

1,600° F.

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1	0.0035	No noticeable increase in carbon.				.....	0.0000
2	0.0070	No noticeable increase in carbon.				.....	0.0000
3	0.0085	No noticeable increase in carbon.				.....	0.0000
4	0.0085	No noticeable increase in carbon.				.....	0.0000
5	0.0145	0.30	.....	.....	.....	0.0132	0.0132
6	0.0225	0.80	.....	.....	0.0060	0.0048	0.0108
7	0.0185	.....	.....	.....	.....	.....	0.0000
8	0.031	0.70	.....	.....	0.0180	0.0060	0.0240
9	0.031	0.70	.....	.....	0.0240	0.0120	0.0360
10	0.073	0.60	.....	.....	0.0084	0.0108	0.0192

1,750° F.

1	0.018	0.50	.....	.....	.....	0.0120	0.0120
2	0.0275	0.80	.....	.....	0.0096	0.0084	0.0180
3	0.0375	0.70	.....	.....	0.0300	0.0072	0.0372
4	0.041	0.60	.....	.....	0.0216	0.0084	0.0300
5	0.036	0.70	.....	.....	0.0132	0.0072	0.0204
6	0.025	0.60	.....	.....	0.0132	0.0132	0.0264
7	0.064	0.70	.....	.....	0.0420	0.0072	0.0492
8	0.0605	0.80	.....	.....	0.0276	0.0084	0.0360
9	0.063	0.70	.....	.....	0.0360	0.0084	0.0444
10	0.0705	0.70	.....	.....	0.0420	0.0072	0.0492

1,900° F.

1	0.022	0.60	.....	.....	0.0084	0.0096	0.0180
2	0.041	0.70	.....	.....	0.0168	0.0168	0.0336
3	0.057	0.90	.....	0.0084	0.0168	0.0276	0.0528
4	0.071	0.85	.....	.....	0.0408	0.0072	0.0480
5	0.080	0.90	.....	0.0168	0.0528	0.0144	0.0840
6	0.080	0.90	.....	0.0048	0.0528	0.0264	0.0840
7	0.085	0.90	.....	0.0132	0.0360	0.0288	0.0780
8	0.092	0.85	.....	.....	0.0576	0.0288	0.0864
9	0.095	0.90	.....	0.0084	0.0576	0.0192	0.0852
10	0.121	0.90	.....	0.0120	0.0624	0.0192	0.0936

Repeated Use at 1,750°.

1st 10	0.0705	0.70	.....	.....	0.0420	0.0072	0.0492
2d 10	0.092	0.90	.....	0.0108	0.0408	0.0228	0.0744
3d 10	0.102	0.90	.....	0.0300	0.0288	0.0156	0.0744
4th 10	0.089	0.90	.....	0.0228	0.0324	0.0072	0.0624

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TABLE XIV.—*Material "H."*

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1,600° F.							
1	0.014	0.90	.....	0.0024	0.0060	0.0024	0.0108
2	0.021	0.90	.....	0.0018	0.0060	0.0036	0.0114
3	0.044	0.90	.....	0.0024	0.0072	0.0072	0.0168
4	0.026	0.80	.....	.....	0.0108	0.0012	0.0120
5	0.035	0.90	.....	0.0024	0.0180	0.0060	0.0264
6	0.046	0.90	.....	0.0108	0.0168	0.0068	0.0444
7	0.053	0.90	.....	0.0120	0.0180	0.0012	0.0312
8	0.062	0.90	.....	0.0120	0.0240	0.0072	0.0432
9	0.071	0.90	.....	0.0156	0.0180	0.0072	0.0408
10	0.073	0.90	....	0.0144	0.0264	0.0072	0.0480
1,750° F.							
1	0.023	0.90	.....	0.0036	0.0072	0.0048	0.0156
2	0.038	0.90	.....	0.0084	0.0108	0.0072	0.0264
3	0.057	0.90	.....	0.0144	0.0156	0.0084	0.0384
4	0.069	0.90	.....	0.0132	0.0168	0.0072	0.0372
5	0.071	0.90	.....	0.0156	0.0204	0.0036	0.0396
6	0.078	0.90	.....	0.0156	0.0216	0.0072	0.0444
7	0.099	0.90	.....	0.0288	0.0216	0.0120	0.0624
8	0.112	1.05	0.0120	0.0288	0.0168	0.0084	0.0660
9	0.126	1.00	0.0180	0.0204	0.0312	0.0048	0.0744
10	0.136	1.10	0.0180	0.0228	0.0240	0.0072	0.0720
1,900° C.							
1	0.042	0.90	.....	0.0048	0.0204	0.0036	0.0288
2	0.073	0.90	.....	0.0240	0.0180	0.0156	0.0576
3	0.096	0.90	.....	0.0252	0.0168	0.0084	0.0504
4	0.116	0.90	.....	0.0384	0.0300	0.0072	0.0756
5	0.136	0.93	0.0084	0.0252	0.0336	0.0204	0.0876
6	0.146	1.00	0.0180	0.0264	0.0300	0.0228	0.0972
7	0.159	0.95	0.0192	0.0288	0.0396	0.0156	0.1032
8	0.167	0.95	0.0276	0.0240	0.0360	0.0084	0.0960
9	0.174	0.90	.....	0.0444	0.0348	0.0228	0.1020
10	0.209	0.92	0.0132	0.0480	0.0552	0.0216	0.1380
Repeated Use at 1,750°.							
1st 10	0.136	1.10	0.0180'	0.0228	0.0240	0.0072	0.0720
2d 10	0.040	0.60	.....	.....	0.0348	0.0132	0.0480
3d 10	0.092	0.90	.....	0.0264	0.0288	0.0132	0.0684
4th 10	0.084	0.90	.....	0.0324	0.0240	0.0084	0.0648

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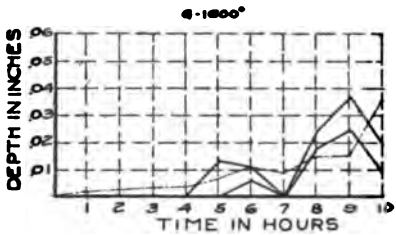

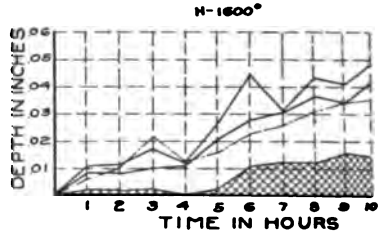

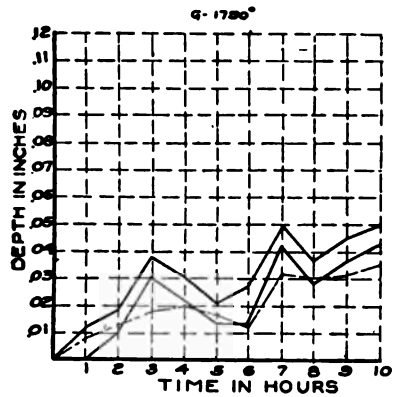

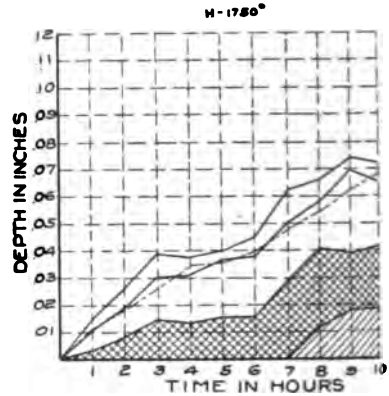

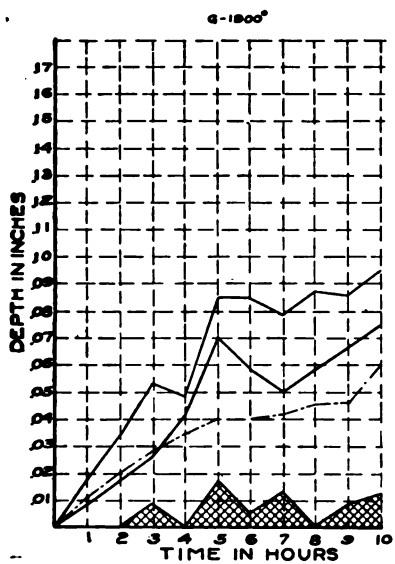

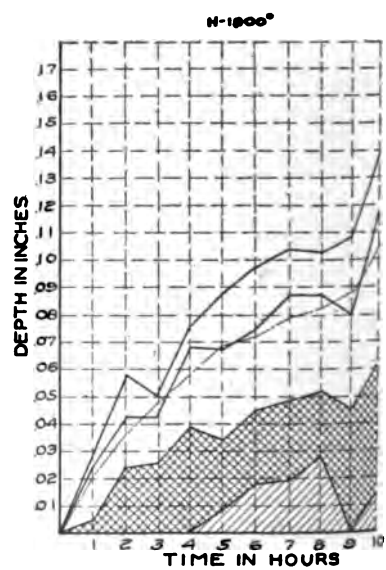

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TABLE XV.—*Material "I."*

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1,600° F.							
1	0.022	0.90	.....	0.0036	0.0048	0.0012	0.0096
2	0.036	0.90	.....	0.0072	0.0060	0.0036	0.0192
3	0.041	0.90	.....	0.0120	0.0060	0.0060	0.0240
4	0.041	0.90	.....	0.0096	0.0048	0.0048	0.0192
5	0.047	1.00	0.0024	0.0060	0.0096	0.0048	0.0228
6	0.062	0.90	0.0120	0.0120	0.0180	0.0012	0.0432
7	0.072	0.95	0.0072	0.0096	0.0144	0.0072	0.0384
8	0.077	1.00	0.0108	0.0036	0.0180	0.0036	0.0360
9	0.058	0.75	.....	.....	0.0444	0.0072	0.0516
10	0.095	1.05	0.0120	0.0108	0.0216	0.0072	0.0516
1,750° F.							
1	0.039	0.90	.....	0.0096	0.0060	0.0024	0.0180
2	0.063	1.10	0.0084	0.0084	0.0096	0.0060	0.0324
3	0.078	1.15	0.0096	0.0180	0.0108	0.0072	0.0456
4	0.083	1.10	0.0108	0.0180	0.0096	0.0060	0.0444
5	0.096	1.00	0.0132	0.0156	0.0168	0.0108	0.0564
6	0.096	1.10	0.0084	0.0204	0.0156	0.0084	0.0528
7	0.083	0.92	0.0084	0.0156	0.0240	0.0060	0.0540
8	0.125	1.15	0.0240	0.0180	0.0180	0.0084	0.0684
9	0.130	0.90	.....	0.0420	0.0300	0.0120	0.0840
10	0.082	0.00	.....	.....	0.0480	0.0120	0.0600
1,900° F.							
1	0.053	0.90	.....	0.0096	0.0204	0.0072	0.0372
2	0.078	0.90	.....	0.0192	0.0216	0.0072	0.0480
3	0.101	0.90	.....	0.0180	0.0288	0.0072	0.0540
4	0.118	0.90	.....	0.0228	0.0408	0.0192	0.0828
5	0.139	0.95	0.0144	0.0228	0.0360	0.0144	0.0876
6	0.147	0.93	0.0084	0.0324	0.0360	0.0240	0.1008
7	0.155	0.90	.....	0.0432	0.0492	0.0060	0.0984
8	0.099	0.20	.....	.....	0.0900	0.0084	0.0984
9	0.096	0.00	.....	.....	0.0780	0.0252	0.1032
10	0.107	0.20	.....	.....	0.0948	0.0288	0.1236

Repeated Use at 1,750° F.

This material would not carbonize after this heat.

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TABLE XVI.—*Material "J."*

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1,600° F.							
1	0.025	0.90	.....	0.0060	0.0036	0.0024	0.0120
2	0.038	0.92	0.0048	0.0036	0.0084	0.0012	0.0180
3	0.048	0.90	.....	0.0096	0.0072	0.0048	0.0216
4	0.049	0.90	.....	0.0108	0.0060	0.0072	0.0240
5	0.054	0.95	0.0024	0.0060	0.0120	0.0060	0.0164
6	0.067	0.95	0.0060	0.0096	0.0120	0.0048	0.0324
7	0.076	0.95	0.0072	0.0096	0.0120	0.0060	0.0348
8	0.083	1.00	0.0084	0.0084	0.0180	0.0072	0.0420
9	0.088	1.00	0.0084	0.0096	0.0216	0.0048	0.0444
10	0.091	1.10	0.0084	0.0180	0.0228	0.0036	0.0528
1,750° F.							
1	0.041	0.90	.....	0.0096	0.0084	0.0012	0.0192
2	0.067	1.15	0.0084	0.0072	0.0168	0.0024	0.0348
3	0.085	1.20	0.0120	0.0108	0.0156	0.0060	0.0444
4	0.098	1.15	0.0180	0.0120	0.0120	0.0060	0.0480
5	0.106	1.15	0.0168	0.0108	0.0108	0.0120	0.0504
6	0.105	1.15	0.0168	0.0120	0.0180	0.0120	0.0588
7	0.105	1.00	0 0132	0.0168	0.0240	0.0072	0.0612
8	0.143	1.20	0.0240	0.0240	0.0360	0.0084	0.0924
9	0.157	1.20	0.0228	0.0216	0.0300	0.0084	0.0828
10	0.162	1.05	0.0264	0.0312	0.0108	0.0156	0.0840
1,900° F.							
1	0.067	1.00	0.0072	0.0168	0.0096	0.0096	0.0432
2	0.097	1.00	0.0108	0.0132	0.0276	0.0096	0.0612
3	0.128	1.10	0.0192	0.0216	0.0216	0.0120	0.0744
4	0.160	1.00	0.0156	0.0312	0.0288	0.0204	0.0960
5	0.173	1.00	0.0180	0.0288	0.0324	0.0180	0.0972
6	0.187	1.10	0.0264	0.0276	0.0360	0.0192	0.1092
7	0.198	1.00	0.0192	0.0444	0.0408	0.0084	0.1128
8	0.203	1.20	0.0312	0.0432	0 0264	0.0084	0.1092
9	0.213	1.10	0.0240	0.0312	0.0528	0.0132	0.1212
10	0.210	1.15	0.0276	0.0384	0.0348	0.0228	0.1236
Repeated Use at 1,750°.							
1st 10	0.162	1.05	0.0264	0.0312	0.0108	0.0156	0.0840
2d 10	0.128	1.00	0.0168	0.0180	0.0264	0.0132	0.0744
3d 10	0.120	0.90	.....	0.0360	0.0288	0.0108	0.0756
4th 10	0.097	0.90	.....	0.0228	0.0384	0.0096	0.0708

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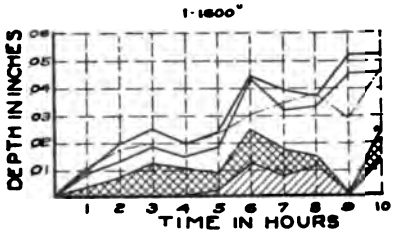

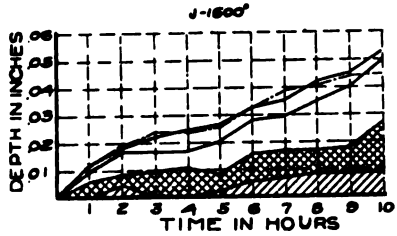

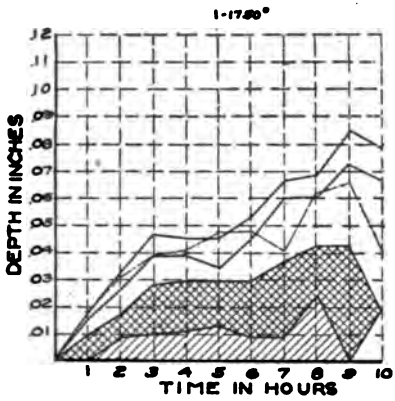

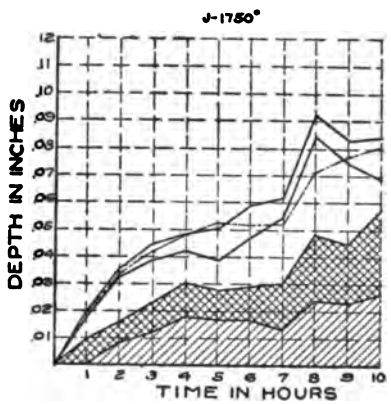

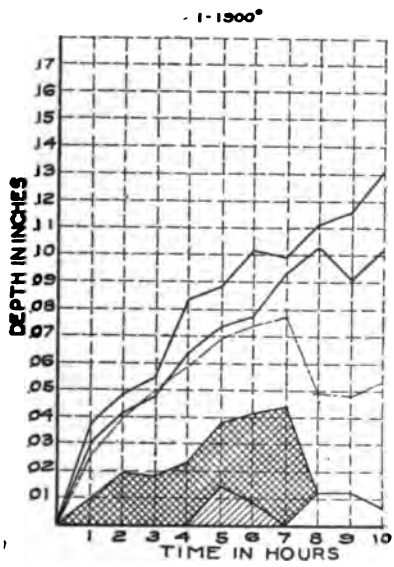

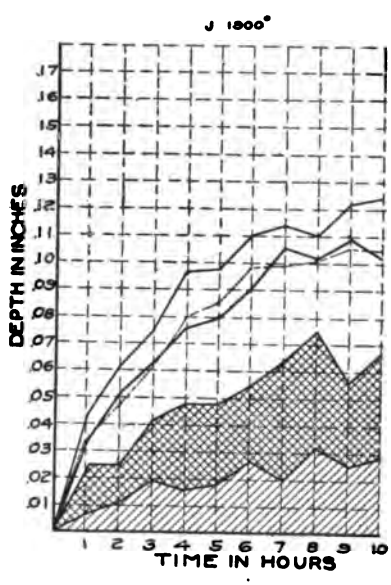

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TABLE XVII.—*Material "K."*

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1,600° F.							
1	0.0115	0.80	.....	.....	0.0060	0.0012	0.0072
2	0.0205	0.90	.....	0.0012	0.0060	0.0048	0.0120
3	0.0235	0.80	.....	.....	0.0108	0.0060	0.0168
4	0.026	0.90	.....	0.0012	0.0120	0.0060	0.0192
5	0.0325	0.90	.....	0.0036	0.0180	0.0060	0.0276
6	0.021	0.60	.....	.....	0.0096	0.0084	0.0180
7	0.0425	0.70	.....	.....	0.0144	0.0120	0.0264
8	0.057	0.80	.....	.....	0.0204	0.0036	0.0240
9	0.0585	0.90	.....	0.0144	0.0204	0.0048	0.0396
10	0.0635	0.90	.....	0.0120	0.0240	0.0072	0.0432
1,750° F.							
1	0.023	0.70	.....	.....	0.0108	0.0024	0.0132
2	0.038	0.90	.....	0.0084	0.0120	0.0060	0.0264
3	0.051	0.90	.....	0.0084	0.0216	0.0060	0.0360
4	0.0615	0.90	.....	0.0120	0.0192	0.0072	0.0384
5	0.067	0.90	.....	0.0120	0.0180	0.0144	0.0444
6	0.0705	0.90	.....	0.0180	0.0192	0.0072	0.0444
7	0.072	0.90	.....	0.0180	0.0288	0.0012	0.0480
8	0.075	0.90	.....	0.0192	0.0228	0.0084	0.0504
9	0.109	0.90	.....	0.0288	0.0300	0.0132	0.0720
10	0.122	1.00	0.0120	0.0228	0.0360	0.0160	0.0768
1,900° F.							
1	0.038	0.90	.....	0.0072	0.0168	0.0108	0.0348
2	0.063	0.90	.....	0.0144	0.0228	0.0192	0.0564
3	0.082	0.90	.....	0.0144	0.0324	0.0168	0.0636
4	0.100	0.90	.....	0.0192	0.0396	0.0072	0.0660
5	0.120	0.90	.....	0.0276	0.0456	0.0084	0.0816
6	0.133	0.90	.....	0.0300	0.0408	0.0192	0.0900
7	0.140	0.95	0.0132	0.0288	0.0360	0 0252	0.1032
8	0.157	1.00	0.0168	0.0204	0.0516	0.0108	0.0996
9	0.165	1.00	0.0144	0.0312	0.0408	0.0252	0.1116
10	0.164	0.90	.....	0.0480	0.0480	0.0144	0.1104
Repeated Use at 1,750°.							
1st 10	0.122	1.00	0.0120	0.0228	0.0360	0.0060	0.0768
2d 10	0.133	0.95	0.0084	0.0276	0.0336	0.0144	0.0840
3d 10	0.046	0.90	.....	0.0108	0.0300	0.0096	0.0504
4th 10	0.096	0.90	.....	0.0228	0.0372	0.0132	0.0732

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TABLE XVIII.—*Material "L."*

(This material was not used at 1,600°.)

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1,750° F.							
1	0.036	0.90	.....	0.0096	0.0096	0.0012	0.0204
2	0.048	0.90	.....	0.0060	0.0120	0.0060	0.0240
3	0.0685	1.00	0.0096	0.0096	0.0144	0.0036	0.0372
4	0.0715	0.90	.....	0.0120	0.0216	0.0084	0.0420
5	0.064	0.90	.....	0.0132	0.0240	0.0060	0.0432
6	0.0865	0.90	.....	0.0276	0.0192	0.0060	0.0528
7	0.089	0.93	0.0060	0.0180	0.0240	0.0108	0.0588
8	0.084	0.95	0.0084	0.0180	0.0224	0.0072	0.0660
9	0.124	1.10	0.0120	0.0300	0.0300	0.0120	0.0840
10	0.1215	0.90	.....	0.0240	0.0360	0.0144	0.0744
1,900° F.							
1	0.044	0.70	.....	.....	0.0156	0.0144	0.0300
2	0.071	0.90	.....	0.0132	0.0216	0.0024	0.0372
3	0.089	0.90	.....	0.0144	0.0324	0.0144	0.0612
4	0.096	0.90	.....	0.0180	0.0456	0.0096	0.0732
5	0.122	0.90	.....	0.0204	0.0480	0.0192	0.0876
6	0.134	0.90	.....	0.0360	0.0396	0.0252	0.1008
7	0.129	0.90	.....	0.0312	0.0480	0.0144	0.0936
8	0.154	0.90	.....	0.0492	0.0348	0.0216	0.1056
9	0.163	0.90	.....	0.0564	0.0328	0.0180	0.0972
10	0.129	0.90	.....	0.0048	0.0660	0.0324	0.1032
Repeated Use at 1,750°.							
1st 10	0.1215	0.90	.....	0.0240	0.0360	0.0144	0.0744
2d 10	0.083	0.90	.....	0.0132	0.0468	0.0108	0.0708
3d 10	0.060	0.70	.....	.....	0.0324	0.0132	0.0456
4th 10	0.047	0.70	.....	.....	0.0360	0.0168	0.0528

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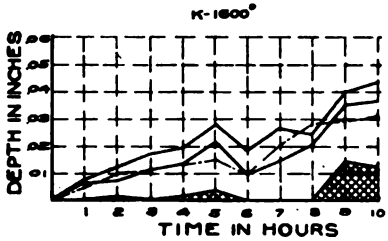

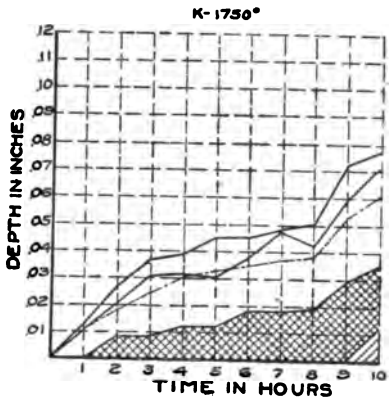

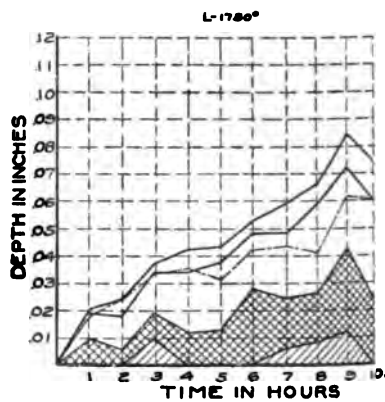

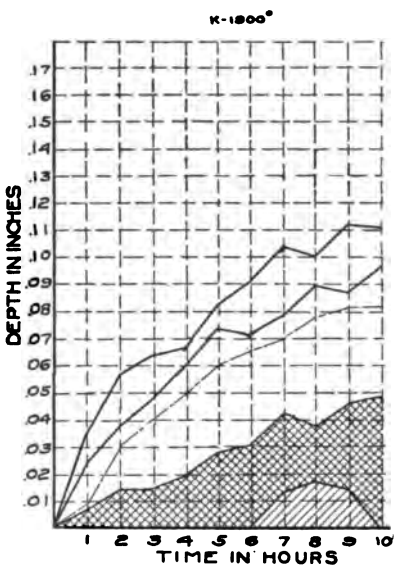

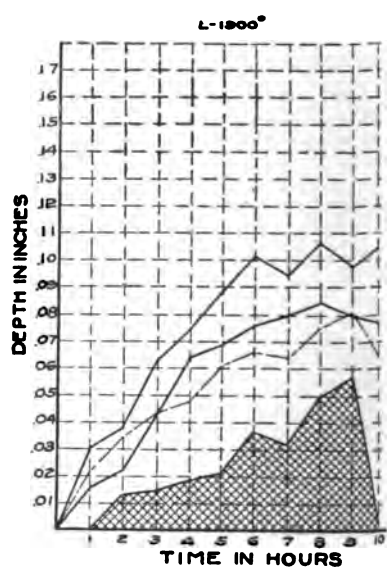

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TABLE XXX.—*Material "X."*

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1,600° F.							
1	0.083	0.90	.....	0.0036	0.0060	0.0012	0.0108
2	0.065	0.95	0.0060	0.0036	0.0069	0.0036	0.0192
3	.....	0.95	0.0036	0.0072	0.0120	0.0060	0.0288
4	.....	0.92	0.0060	0.0048	0.0120	0.0048	0.0276
5	0.080	1.00	0.0066	0.0072	0.0120	0.0120	0.0378
6	0.080	1.10	0.0096	0.0060	0.0120	0.0144	0.0420
7	.....	1.00	0.0084	0.0096	0.0144	0.0048	0.0372
8	.....	1.10	0.0060	0.0060	0.0180	0.0108	0.0408
9	.....	0.95	0.0060	0.0072	0.0180	0.0096	0.0408
10	.....	0.90	.....	0.0144	0.0240	0.0060	0.0444
1,750° F.							
1	0.051	0.90	.....	0.0024	0.0084	0.0072	0.0180
2	0.048	0.95	0.0072	0.0060	0.0072	0.0060	0.0264
3	0.073	1.00	0.0084	0.0096	0.0168	0.0120	0.0468
4	.....	0.80	.....	.....	0.0396	0.0144	0.0540
5	0.103	0.93	0.0024	0.0252	0.0228	0.0144	0.0648
6	.....	1.10	0.0180	0.0108	0.0276	0.0072	0.0636
7	0.126	0.95	0.0156	0.0264	0.0228	0.0084	0.0732
8	0.110	1.05	0.0132	0.0216	0.0324	0.0120	0.0792
9	.....	0.95	0.0132	0.0288	0.0276	0.0132	0.0828
10	0.139	0.95	0.0192	0.0192	1.0372	0.0144	0.0900
1,900° F.							
1	0.068	0.93	0.0036	0.0096	0.0144	0.0084	0.0360
2	0.081	0.90	.....	0.0216	0.0240	0.0228	0.0684
3	0.089	0.90	.....	0.0132	0.0324	0.0192	0.0648
4	0.108	1.00	0.0120	0.0156	0.0312	0.0132	0.0720
5	0.117	0.90	.....	0.0360	0.0408	0.0252	0.1020
6	0.122	0.85	.....	.....	0.0780	0.0336	0.1116
7	0.143	0.90	.....	0.0360	0.0540	0.0312	0.1212
8	0.145	0.90	.....	0.0468	0.0540	0.0276	0.1284
9	0.161	0.90	.....	0.0540	0.0444	0.0240	0.1224
10	0.166	0.90	.....	0.0408	0.0660	0.0216	0.1284
Repeated Use at 1,750°.							
1st 10	0.139	0.95	0.0192	0.0192	0.0372	0.0144	0.0900
2d 10	0.187	0.90	.....	0.0132	0.0456	0.0036	0.0624
3d 10	0.016	0.70	.....	.....	0.0456	0.0192	0.0648
4th 10	Lost	weight.	Would	not car	bonize	after this	heat.

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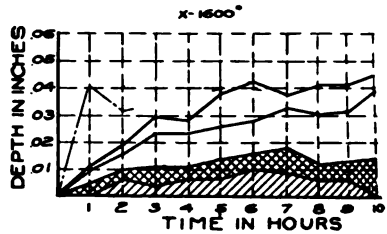

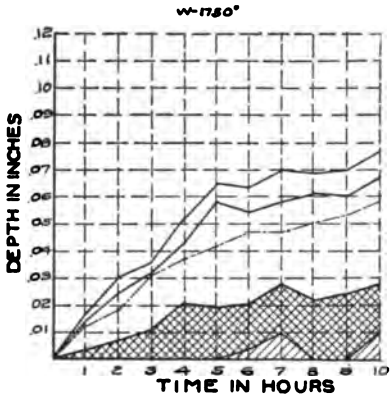

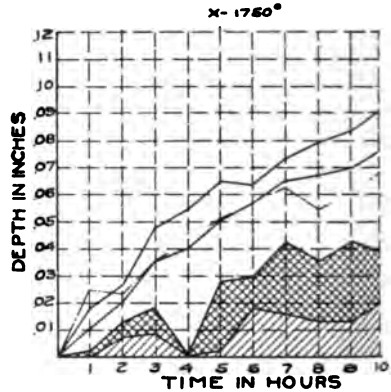

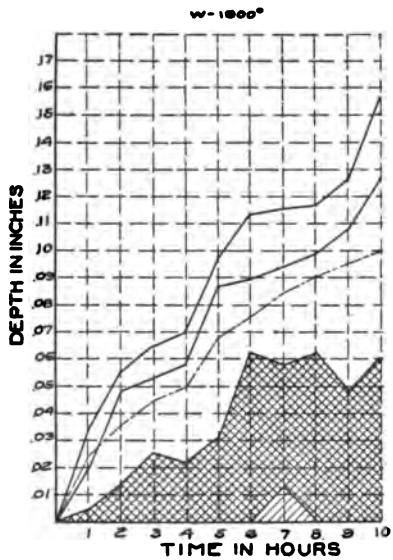

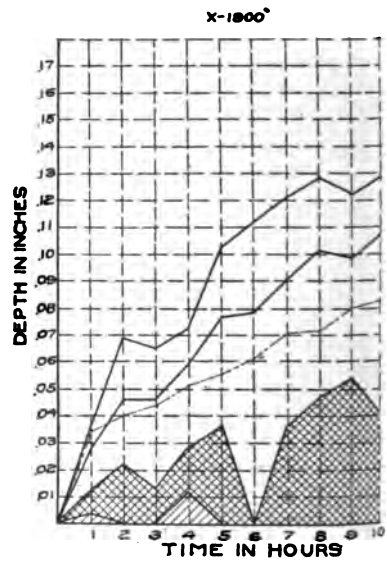

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TABLE XXXI.—*Material "Y."*

Hours.	Increase in Weight.	Surface Carbon.	Depth of Carbon.				
			Over 0.90 Per Cent.	0.90 Per Cent.	0.90 to 0.50 Per Cent.	0.50 to 0.20 Per Cent.	Total.
	Grams.	Per Cent.	Inches.	Inches.	Inches.	Inches.	Inches.
1,600° F.							
1	0.017	0.90	.....	0.0060	0.0036	0.0030	0.0126
2	.....	0.80	.....	.....	0.0180	0.0012	0.0192
3	.....	0.92	0.0024	0.0060	0.0120	0.0060	0.0264
4	.....	0.92	0.0048	0.0060	0.0156	0.0012	0.0276
5	0.071	0.95	0.0084	0.0060	0.0132	0.0036	0.0312
6	.....	0.91	0.0084	0.0108	0.0156	0.0060	0.0408
7	.....	0.95	0.0060	0.0036	0.0180	0.0084	0.0360
8	.....	1.00	0.0120	0.0132	0.0156	0.0072	0.0480
9	.....	0.95	0.0072	0.0072	0.0180	0.0084	0.0408
10	.....	0.92	0.0024	0.0144	0.0240	0.0060	0.0468
1,750° F.							
1	0.045	0.92	0.0048	0.0060	0.0048	0.0024	0.0180
2	0.064	0.90	.....	0.0144	0.0096	0.0084	0.0324
3	0.063	1.15	0.0132	0.0084	0.0108	0.0072	0.0396
4	0.107	1.10	0.0132	0.0084	0.0204	0.0060	0.0480
5	0.137	1.15	0.0240	0.0144	0.0108	0.0168	0.0660
6	0.136	1.05	0.0192	0.0156	0.0204	0.0180	0.0732
7	0.122	1.00	0.0156	0.0216	0.0228	0.0072	0.0672
8	0.141	0.95	0.0144	0.0192	0.0156	0.0168	0.0660
9	0.156	1.00	0.0240	0.0108	0.0372	0.0048	0.0768
10	0.175	1.15	0.0240	0.0168	0.0336	0.0204	0.0948
1,900° F.							
1	0.069	0.90	.....	0.0156	0.0132	0.0096	0.0384
2	0.091	0.90	.....	0.0228	0.0312	0.0048	0.0588
3	0.106	0.90	.....	0.0336	0.0300	0.0084	0.0720
4	0.108	0.90	.....	0.0324	0.0372	0.0120	0.0816
5	0.118	0.85	.....	.....	0.0672	0.0156	0.0828
6	0.136	0.90	.....	0.0084	0.0684	0.0192	0.0960
7	0.147	0.90	.....	0.0372	0.0564	0.0108	0.1044
8	0.140	0.90	.....	0.0252	0.0888	0.0300	0.1440
9	0.156	0.90	.....	0.0504	0.0588	0.0192	0.1284
10	0.159	0.80	.....	.....	0.0984	0.0360	0.1344
Repeated Use at 1,750°.							
1st 10	0.175	1.15	0.0240	0.0168	0.0336	0.0204	0.0948
2d 10	0.098	0.95	0.0036	0.0264	0.0228	0.0168	0.0696
3d 10	0.003	0.90	.....	0.0204	0.0384	0.0060	0.0708
4th 10	0.008	0.90	.....	0.0024	0.0324	0.0180	0.0528

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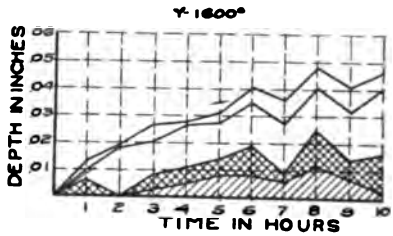

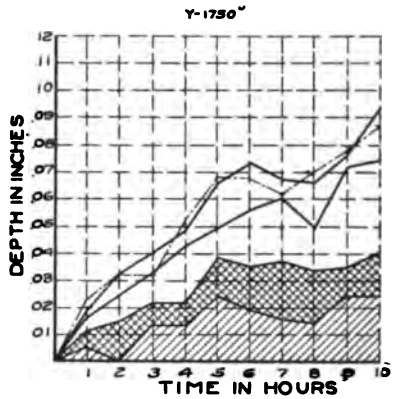

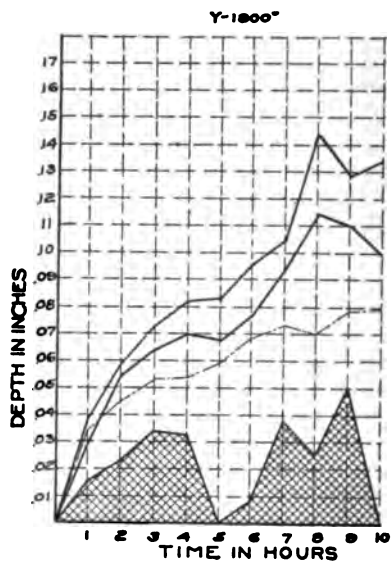

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Discussion.

H. D. Hibbard, Plainfield, N. J.:—Were any of these carbonizing-materials gaseous?

Mr. Abbott:—None of them were.

Mr. Hibbard:—It is regrettable that the gases were not included. From some indications gas carbonizing is leading at present, and I would like to know the performance of gaseous as compared with solid carbonizers.

Mr. Abbott:—Straight gases were not included because the materials which I used gave up the various gases which are being used in gas carbonizing. In other words, I generated in the materials the gases which are sometimes used as compressed gases.

Mr. Hibbard:—It is considered uncertain what gases and what amount of them are given out, and the temperatures at which they are set free from these materials. I have heard it claimed that the temperatures at which case-hardening proceeds rapidly are above those at which the gases are given out, and so by the time that the gases are pretty well expelled from the material, case-hardening is hardly ready to begin, and therefore the results have been non-uniform and unsatisfactory.

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Note on the Case-Hardening of Special Steels.

BY ALBERT SAUVEUR* AND G. A. REINHARDT,† CAMBRIDGE, MASS.

(Cleveland Meeting, October, 1912.)

ALTHOUGH many metallurgists know that some pearlitic special steels can be made troostitic, martensitic, and even austenitic, without quenching, and, therefore, without exposing them to the dangers of the quenching-bath, the practical significance of such possibility does not seem to have received the attention it deserves. Let us recall briefly the mechanism of the transformation alluded to.

In Fig. 1 (after Guillet) is shown the constitutional diagram of nickel-steel; that is, of alloys of iron, carbon, and nickel. Any steel the composition of which falls within the area *LPO* is pearlitic after slow cooling; any steel the composition of which is represented by a point within the area *MLO* is martensitic, and any steel within the area *MNO* is austenitic. Starting with pearlitic steel, then by increasing the carbon-content or the nickel-content, or both, the metal becomes martensitic and eventually austenitic; that is, its composition crosses first the boundary-line *LO* and then the line *MO*—and it remains martensitic or austenitic after slow (air) cooling.[1] We have in this way imparted to the metal without quenching it the properties of hardened carbon-steel. In other words, the steel is now self-hardening. Nickel cannot be introduced into solid steel, but by the well-known process of case-hardening it

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[1] The boundaries between the various zones are not, as a matter of fact, as sharp as indicated in the diagram, because pearlite is not transformed abruptly and in its entirety into martensite when a certain critical composition is attained, nor is martensite suddenly and completely converted into austenite. Greater refinement in the construction of the diagram would undoubtedly result in the introduction of troosto-pearlitic, troostitic, and troosto-martensitic zones between the pearlitic and martensitic areas and of a martenso-austenitic zone between the martensitic and austenitic areas. For the purpose of this note, however, the existence of these transition-conditions may be ignored.

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should be possible to cause pearlitic nickel-steel to absorb the necessary amount of carbon to become martensitic, or even, if desired, austenitic. It should therefore be possible to manufacture case-hardened objects of nickel-steel and of some other special steels having pearlitic, and consequently soft and tough, cores, and troosto-martensitic, martensitic, martenso-austenitic or austenitic cases, without recourse to the quenching-bath. In Fig. 1, for instance, the point A represents a steel containing 0.25 per cent. of carbon and 3 per cent. of nickel. By increasing the carbon-content to 1.2 per cent. its composition shifts from A to B and the metal becomes martensitic. Increasing its car-

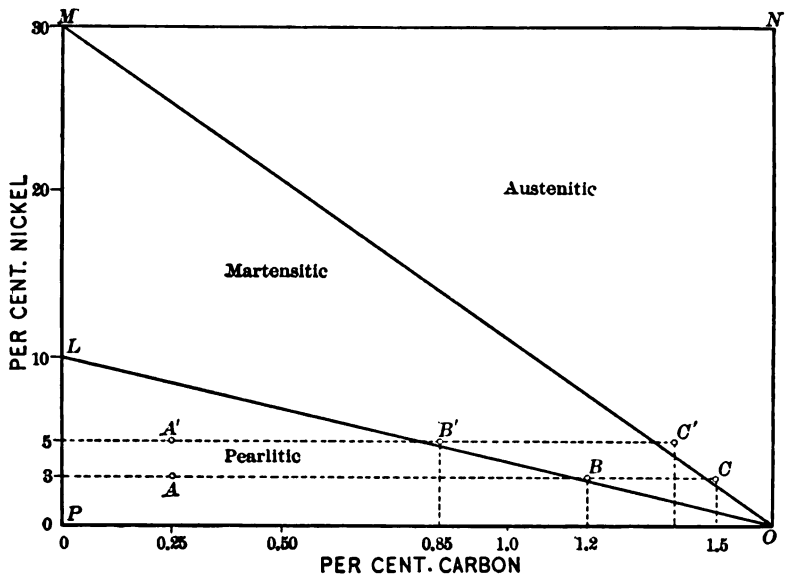

FIG. 1.—CONSTITUTIONAL DIAGRAM OF NICKEL-STEEL (GUILLET).

bon to 1.5 per cent. would make it austenitic, since its composition would now be represented by the point C. A pearlitic steel containing the same amount of carbon (0.25 per cent.), but more nickel, say 5 per cent., would require a smaller addition of carbon to become martensitic or austenitic, since by shifting its composition from A' to B' when it contains 0.85 per cent. of carbon it fails within the martensitic area. It is evident that the nearer the composition of pearlitic steel to the boundary-line LO the less carbon will it require to make it martensitic.

To test the practicality of the manufacture of case-hardened

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articles of the description referred to, a number of nickel-steels of varying compositions were subjected to the case-hardening treatment, and no difficulty was experienced in producing martensitic cases and even, in the presence of sufficient nickel, austenitic cases.

It will suffice for the purpose of this note to describe briefly a few of the tests performed.

Steel No. 3.

Origin.—Nickel-steel supplied by the Peerless Motor Car Co., of Cleveland, Ohio.

Shape.—1-in. hexagonal bar.

Chemical Composition.—Ni, 3.44; C, 0.176; P, 0.011; S, 0.026; Mn, 0.48 per cent.

Treatment A.—Annealed for 3 hr. at a temperature of 1,000° C., surrounded by a mixture composed of 40 per cent. of barium carbonate and 60 per cent. of wood-charcoal, and cooled in air.

Results.—Approximate thickness of case, 1.3 mm. Approximate thickness of martensitic portion of case, 0.5 mm. The balance of the case was troosto-martensitic, troostitic, and troosto-pearlitic.

The microstructure of the case is shown in Figs. 2 and 3, magnified respectively 50 and 100 diameters. The purely martensitic character of the shell of the case will be noted, as well as the occurrence of troostite and its gradual increase. In Fig. 2 the wide black band is pure troostite. The martensitic grains owe their polyhedral form to the original austenitic grains from which they are derived. Some of the grains are surrounded by troostitic membranes.

Treatment B.—Same as treatment A, but lasting 5 hr.

Results.—Case of the same character, but measuring 1.7 mm. in thickness and the martensitic portion 0.9 mm.

Treatment C.—Same as treatment B, but the steel was allowed to cool slowly in the case-hardening furnace after the operation.

Results.—Approximate thickness of case, 1.9 mm. The case consisted of sorbitic grains surrounded by free cementite and containing many cementite needles, as shown in Fig. 4.

By referring to Fig. 1 it will be seen that in the presence

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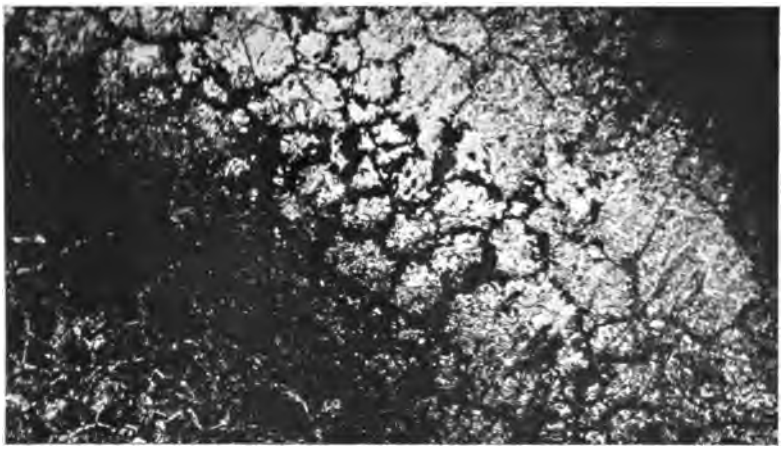

FIG. 2.—NICKEL-STEEL. Ni, 3.44; C, 0.176 PER CENT. CASE-HARDENED STRUCTURE OF CASE. MAGNIFIED 50 DIAMETERS.

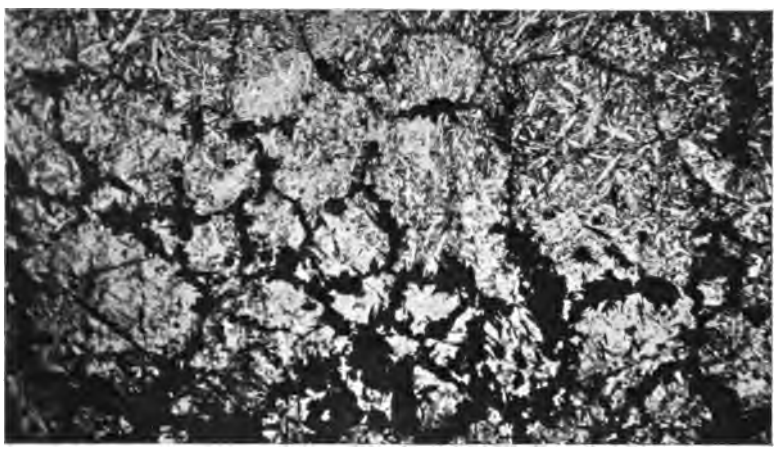

FIG. 3.—SAME STEEL AS IN FIG. 2. SAME TREATMENT. MAGNIFIED 100 DIAMETERS.

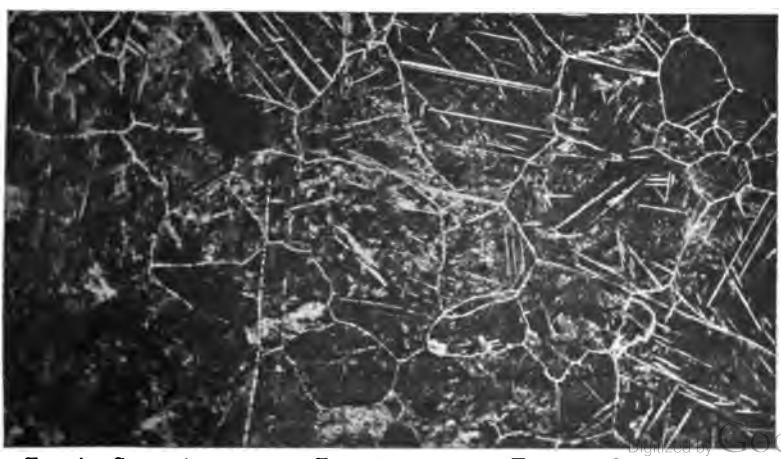

FIG. 4.—SAME STEEL AS IN FIGS. 2 AND 3, BUT FURNACE-COOLED AFTER CASE-HARDENING. MAGNIFIED 100 DIAMETERS.

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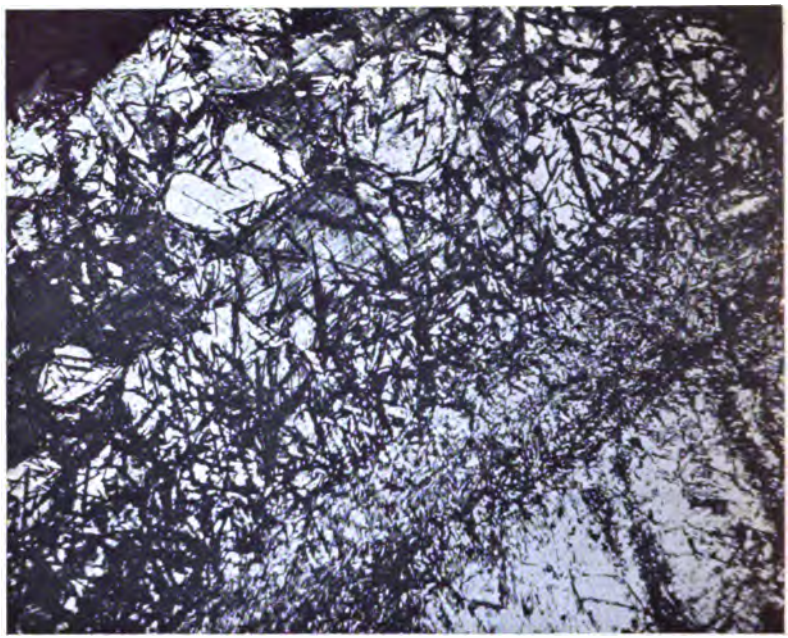

FIG. 5.—NICKEL-STEEL. Ni, 4.86; C, 0.115 PER CENT. CASE-HARDENED. STRUCTURE OF CASE. MAGNIFIED 100 DIAMETERS.

FIG. 6.—SAME STEEL AS IN FIG. 5. SAME TREATMENT. MAGNIFIED 300 DIAMETERS.

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of 3.5 per cent. of nickel some 1.2 per cent. of carbon is needed in order that the steel may be martensitic after air-cooling. It will be evident from the structure of the slowly-cooled sample (Fig. 4) that at least that amount of carbon had been introduced.

Conclusions.—Steel containing some 3.5 per cent. of nickel can be made martensitic by case-hardening followed by air-cooling. Slow cooling in the furnace produces a sorbito-cementitic case.

Other Treatments.—The same steel was case-hardened, using crushed bone, but the results were much less satisfactory.

Steel No. 5.

Origin.—Nickel-steel supplied by the Peerless Motor Car Co., of Cleveland, Ohio.

Shape.—1.5-in. round bars.

Chemical Composition.—Ni, 4.86; C, 0.115; P, 0.010; S, 0.037; Mn, 0.50 per cent.

Treatment A.—Annealed for 5 hr. at a temperature of 1,000° C., surrounded by a mixture composed of 40 per cent. of barium carbonate and 60 per cent. of wood-charcoal, and cooled in air.

Results.—Approximate thickness of case, 2.5 mm. Approximate thickness of austenitic portion of case, 0.5 mm. Approximate thickness of martensitic portion, 1.3 mm. The balance of the case was troosto-martensitic, troostitic, and troosto-pearlitic.

The microstructure of the case is shown in Figs. 5 and 6, magnified 100 and 300 diameters respectively. The white constituent is austenite, the dark needles martensite.

Treatment B.—Same as treatment A with the exception that the bar was allowed to cool slowly with the furnace after the case-hardening operation.

Results.—A case about 2 mm. thick was produced consisting of large grains of sorbite and much free cementite.

By referring to Fig. 1 it will be seen that in the presence of 5 per cent. of nickel or thereabout some 0.9 per cent. of carbon should theoretically suffice to produce a martensitic case, while 1.30 per cent. of carbon or more should make it austenitic. The results obtained in these case-hardening tests agree satisfactorily with the diagram.

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Conclusions.—In case-hardening nickel-steel containing about 5 per cent. of nickel, austenitic or martensitic cases can readily be produced through proper regulation of the length of the operation and by cooling in air the case-hardened articles. Slow cooling in the furnace produces sorbito-cementitic cases.

Treatment of Sorbito-Cementitic Cases.—By reheating to 1,000° C. a steel with a sorbito-cementitic case (because of slow cooling in the furnace) and cooling in air, the case became martensitic. This suggests the advisability of cooling in the furnace after case-hardening, producing a sorbito-cementitic case, and then reheating to 800° C. and cooling in air, thereby securing a finer martensitic case than by cooling in air from the case-hardening temperature.

Commercial Applications.

The possible commercial applications of a case-hardening operation yielding martensitic cases without recourse to the quenching-bath are obvious. It should be possible to case-harden finished pieces of machinery, gears for instance, of low-carbon nickel-steel, producing hard wearing-surfaces which should be free from the cracks, distortions, and internal strains so frequently present in quenched steel.

We are indebted to the Peerless Motor Car Co. for supplying the steel used in these tests, and to R. R. Abbott, metallurgist of the company, for his interest in the work and valuable suggestions.

Discussion.

Robert R. Abbott, Cleveland, Ohio:—The possibilities of the commercial application of the facts brought out in this paper have been the subject of many experiments by me during the past four years. It will be well to call attention to several facts having a commercial bearing upon the problem:

(1) The lower critical temperature (Ac_1) of a 3.5 per cent. nickel-steel of low carbon (say 0.20 per cent.) content is about 710° C. This is not lowered to any extent by increasing its carbon content by carbonizing. By this I mean that a steel with Ac_1 occurring at 710°, must, after carbonizing, be heated above this temperature before quenching in order to make it hard. This is not contrary to Guillet's diagram, although at

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first thought it may seem so, due to a confusion of the Ac_1 and the Ar_1 points. His diagram merely gives the physical condition of the steel upon slow cooling. Now a steel of 1.20 per cent. of carbon and 3 per cent. of nickel (point *B* of the diagram) may be martensitic upon slow cooling from three causes:

(*a*) Its Ar_1 point is below the normal atmospheric temperatures.

(*b*) The time element is so important that the change from martensite to pearlite and cementite does not have time to occur even with this slower rate of cooling. We of course know that nickel has the tendency of retarding this change.

(*c*) A combination of the above two causes.

(2) With increasing nickel content the ability of steel to absorb carbon decreases.

(3) Many carbonized nickel-steels show a martensitic structure upon air-cooling, and yet are so soft that they are of no commercial value.

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Notes on Ruff's Carbon-Iron Equilibrium Diagram.*

BY HENRY M. HOWE, NEW YORK, N. Y.†

(Cleveland Meeting, October, 1912.)

PROFESSOR RUFF'S most illuminating paper[1] describing his extremely valuable investigation of the carbon-iron equilibrium diagram assigns definite temperatures to certain very important lines in the diagram, viz.: the eutectic temperature *EB* (Fig. 1) and the "near-eutectic" temperature *E'B'*, but without indicating clearly enough the reasoning which warrants these assignments. These temperatures are of such primary importance that it is well to indicate the apparent gaps in the reasoning, in the hope that they may be filled in.

In his diagram, Fig. 1, the point *X* and the lines *B'D'HI* represent the experimental results reached by Ruff and Goecke;[2] *AB*, *AEB*, and *SE* represent the liquidus, the solidus, and the solubility of the carbon in solid austenite in the absence of graphite according to Gutowsky,[3] *AB* being also in accordance with Spiecker's data;[4] *S'E'* the solubility of carbon in solid austenite in the presence of graphite according to Ruer and Iljin;[5] *GS* the line *A3* after Goerens and Meyer;[6] and *EBC* the true eutectic line as inferred by Prof. Ruff from Gutowsky's data. *B'D'* is the carbon content found in cast-iron which had apparently saturated itself with carbon in the presence of the graphite of the enclosing graphite crucibles in which the determinations were made. From its straightness and its sharp inflection near 6.67 per cent. of carbon, the carbon content of

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[1] *Metallurgie*, vol. viii., pp. 456 to 464 and 497 to 508 (1911).

[2] *Idem*, vol. viii., pp. 417 to 421 (1911).

[3] *Idem*, vol. vi., pp. 731 to 743 (1909).

[4] *Idem*, vol. vi., p. 524 (Wüst) (1909).

[5] *Idem*, vol. viii., pp. 97 to 101 (1911).

[6] *Idem*, vol. vii., pp. 307 to 312 (1910).

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cementite, Fe_3C, he infers that the carbon exists in the molten iron almost solely in the form of cementite in this range of temperature. In a molten alloy above $B'D'$ the dissolved cementite is stable. If it passes below $B'D'$ all cementite then in excess of $B'D'$ is thereby made metastable and tends to precipi-

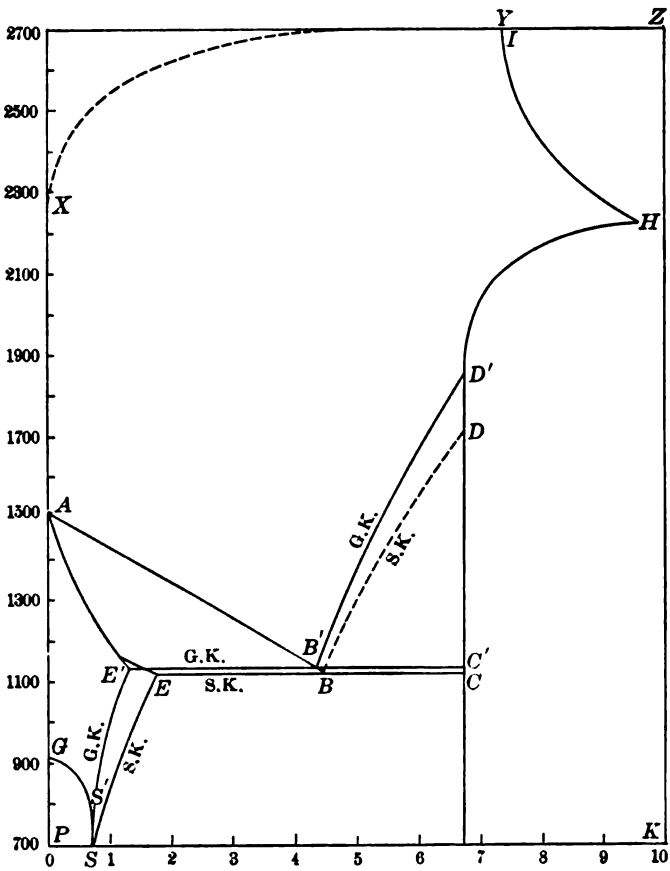

FIG. 1.—RUFF'S CARBON-IRON EQUILIBRIUM DIAGRAM. THE *GK* LINES REPRESENT EQUILIBRIUM WITH GRAPHITE AS BODENKOERPER, THE *SK* LINES WITH CEMENTITE AS BODENKOERPER.

tate, and that which so precipitates graphitizes promptly. *BD*, not determined experimentally, represents the true solubility of cementite in molten iron, a line on reaching which the precipitation of cementite accelerates.

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The later results of Wittorff,[7] not yet readily accessible, are at variance in many respects with Ruff's for high carbon contents.

Let us now review the conditions underlying the matter which I discuss, in the form of 20 propositions in which I attempt to give the premises from which Prof. Ruff reaches his conclusions. My general contention is that, granting these propositions, the steps in Prof. Ruff's reasoning ought to be stated more fully in order to carry conviction. Indeed, if we make in proposition (15) the modification explained in (32), and this seems a permissible one, then the data seem explicable in the simpler way set forth in (28) to (31). Further, the Gutowsky data on which Prof. Ruff bases his calculations seem insufficient in view of the capriciousness of the phenomena and the difficulty of reconciling some of them with our natural conceptions. My purpose thus is to bring the matter up for discussion, and only as a basis for discussion do I offer my tentative explanation. The matter enclosed in parentheses in the text of these twenty propositions represents ideas of my own for which Professor Ruff is not responsible.

General Outline.—(1) to (21), pp. 432 to 442, general outline of Ruff's premises.

(22), p. 443, general meaning of his diagram.

(23) to (26), pp. 443 to 448 Ruff on the eutectic temperature.

(26A) to (35), pp. 448 to 456, other positions of the eutectic temperature.

(36) p. 457. Capriciousness of the rate of graphitization and of the arrest and recalescence temperatures.

(37), p. 462. Graphitization goes on at temperatures below those admitted by Prof. Ruff.

(38) p. 465. How dominant is the bodenkoerper?

(39) p. 467. Conclusion.

(1) Carbon when dissolved either in the molten iron between say 1,800° and 1,135°, or in the austenitic solid iron above the bottom of the transformation range, A1 (say 725°), exists almost solely as tri-ferrous carbide, Fe_3C, called cementite. (P. 460, ¶ 2.) These and the following page numbers refer to Prof.

[7] *Journal of the Russian Physico-Chemical Society*, vol. xliii., pp. 1613 to 1690 (1911); through *Chemisches Zentralblatt*, part 1, pp. 1091 to 1094 (1912); *Journal of the Society of Chemical Industry*, vol. xxxi., p. 388 (1912).

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Ruff's paper, *Metallurgie*, vol. viii., 1911. The section numbers in parentheses refer to the present paper.

(2) Between these temperatures, say 725° and 1,800°, the quantity of carbon which exists in the elemental state in solution in the iron is so small that the true graphite-iron diagram plays only an indirect part. (P. 458.) (Some might be inclined to regard it as hardly more than a metaphysical abstraction.) The diagram which iron in solidifying actually follows is the cementite-austenite diagram, though this too is in a sense a double diagram, one set of lines representing equilibrium when graphite is the "bodenkoerper," and the other set representing true solubility, *e. g.*, when cementite is the bodenkoerper, a term the meaning of which is set forth in Appendix II.

(3) The line $B'D'$, experimentally determined by Ruff and Goecke,[8] represents the quantity of carbon as cementite which the molten iron holds when in equilibrium with solid graphite as bodenkoerper (pp. 458, 459, 460) and the line $S'E'$ (p. 456), adopted from Ruer and Iljin,[9] represents that which the solid austenite holds when in equilibrium with graphite as bodenkoerper, as in gray cast-iron. The line BD (p. 500) represents the true solubility of cementite in molten iron, and SE (p. 456) its true solubility in solid austenite, *i.e.*, the quantity which the molten iron and the solid solution or austenite hold in equilibrium with solid cementite. This line is important because on reaching it the precipitation of cementite accelerates; and its importance does not rest on any assumption as to the ability of free cementite to resist graphitization at these very high temperatures. (In (38) facts are given which are not at first reconciled easily with this idea that the nature of the bodenkoerper is of the first importance in determining whether the dissolved carbon content shall follow SE or $S'E'$.)

(4) From (2) it follows that, when the dissolved carbon content or concentration of the molten is that represented by B (Figs. 1 and 2), or in undercooling by a point on BB_4 (Figs. 2 and 3), the dissolved carbon content of the solid austenite then depositing from that molten should be represented by the point horizontally opposite on SE; and that when the dissolved carbon content of the molten is represented by B', or in undercooling by

[8] *Metallurgie*, vol. viii., pp. 417 to 421 (1911).

[9] *Idem*, vol. viii., pp. 97 to 101 (1911).

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a point on $B'Q$, the dissolved carbon content of the solid austenite then depositing should be represented by the point horizontally opposite on $S'E'$. Conversely, the fact that the dissolved carbon content of the austenite depositing is represented by a point on SE or $S'E'$, as the case may be, implies that the

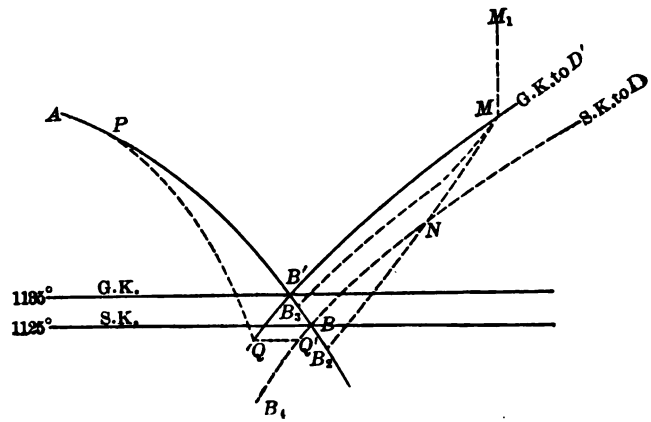

FIG. 2.—THE COURSE OF THE MOLTEN DURING SOLIDIFICATION, APPROXIMATELY AFTER RUFF.

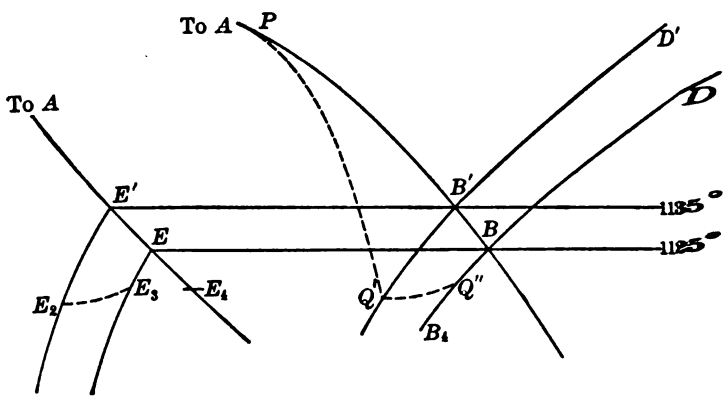

FIG. 3.—THE COURSE OF THE MOLTEN AND OF THE SOLID DURING SOLIDIFICATION, APPROXIMATELY AFTER RUFF.

dissolved carbon content of the molten from which that austenite is depositing should be represented by the point horizontally opposite on BB_4 or B'. (The conception that in undercooling the course of the depositing layers is along $E'E$ need not here disturb us. For instance if, in undercooling, the

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course of the molten down to the trough of the recalescence is $PQ'Q''$ (Fig. 3), and if the course of the solidifying layers is correspondingly $E'EE_4$, then for instance at E_4, when the molten reaches SK (see (5)) at Q'', the dissolved carbon content of the austenite is E_3, the point horizontally opposite on SE, and the difference, E_4E_3, is represented by free cementite, which graphitizes rapidly at this temperature. And in general while the molten is crossing from Q' on GK to Q'' on SK, the austenite of the depositing layers is crossing from E_2 to E_3 on SE.)

(5) GK and SK. For brevity $B'D'$ and $S'E'$, and $B'Q'$ the prolongation of $B'D'$ in undercooling, and the horizontal $E'B'$ which connects the points at which they end in equilibrium he calls GK; and BD and SE, and BB_4 the prolongation of BD in undercooling, and the horizontal true eutectic line EB which connects E and B in equilibrium he calls SK (*Gleichgewichtskonzentration* and *Saettigungskonzentration* respectively, pp. 458 and 463).

Nomenclature.—Ruff and some others use the unprimed letters SE for the part of SK which lies below the freezing range, but the primed letters $B'D'$ for that part of it which includes the freezing range and the molten, using $S'E'$ for GK in the solid and BD for GK in the molten. But because it is simpler to represent the whole of one line by means of unprimed and the whole of the other line by primed letters, and because the most firmly fixed and also the earliest usage is that which associates unprimed SE with SK, I represent SK for the solid and for the molten respectively by SE and BD, and GK by $S'E'$ and $B'D'$.

In the same way the GK or "near-eutectoid" temperature at which $S'E'$ cuts GOS may be called $A'1$.

(6) All carbon dissolved as cementite up to GK is strictly stable (p. 463).

(7) Carbon dissolved as cementite in excess of GK and up to SK is metastable, and tends to precipitate and graphitize, $Fe_3C = 3Fe + Gr$ (p. 463). (It is simplest to assume that the graphitization is confined to cementite which actually precipitates. See p. 462, ¶ 4, lines 12 to 14).

(8) EB, the horizontal which connects the points at which SE and BD end in equilibrium, is the true metastable or cementite-austenite eutectic temperature (p. 463 and p. 498, line 6).

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(9) $E'B'$, the horizontal which connects the points at which $S'E'$ and $B'D'$ end when in equilibrium, is no true eutectic line, but like the lines which it connects merely the temperature on cooling past which cementite changes progressively from being dissolved in perfect stability to being dissolved in metastability, tending to precipitate out and if so precipitated to graphitize. (For brevity it might be called a "near-eutectic" line) (p. 462).

(10) The carbon content of E', *i. e.*, the solubility of carbon in solid austenite when graphite is bodenkoerper at temperature $E'B'$, is taken from Ruer and Iljin as 1.3 per cent.[10] (p. 456).

(11) The carbon content of E, *i. e.*, the true solubility of cementite in solid austenite at temperature EB, cementite being bodenkoerper, is taken from Gutowsky as 1.7 per cent.[11] (p. 456).

(12) Carbon dissolved in excess of SK precipitates promptly as cementite (p. 498). (I find no direct enunciation of this, but the reasoning evidently assumes this.)

Thus, there are four conditions of cementite: (A) stable up to GK; (B) metastable and tending to precipitate, GK to SK; (C) in excess of SK (labile?) precipitating promptly; and (D) that actually precipitated, whether from (B) or (C).

(13) Cementite actually precipitated graphitizes with great rapidity at temperatures above 1,135°. Professor Ruff's language leaves on one the impression that this graphitization is well nigh instantaneous. But Hanemann's results[12] show that, rapid as it is, it can yet be restrained by a rapid enough cooling. Wittorff's language also implies that the carbide which here precipitates can be retained ungraphitized, though as to the nature of this carbide he and Ruff disagree.[13]

(14) Beginning at 1,135°, with further fall of temperature graphitization is retarded very greatly, so that, according to Prof. Ruff's calculations, whereas at 1,135° the time occupied by the graphitization of 50 per cent. of the cementite precipitated in ex-

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[10] *Metallurgie*, vol. viii., p. 99 (1911).

[11] *Idem*, vol. vi., p. 739 (1909.)

[12] *Stahl und Eisen*, vol. xxxi., pp. 333–336 (1911).

[13] *Journal of the Russian Physico-Chemical Society*, vol. xliii., pp. 1613 to 1690 (1911); through *Chemisches Zentralblatt*, part 1, pp. 1091 to 1094 (1912); *Journal of the Society of Chemical Industry*, vol. xxxi., p. 388 (1912).

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cess of GK is 2 sec., this time doubles with every 10° fall of temperature, becoming 32 sec. at 1,085° (or 1,095°) (pp. 502 and 507). These numbers refer to graphitization during solidification, when the cementite crystals are still bathed with molten iron. The rate of graphitization within solid iron Prof. Ruff seems to regard as negligibly slow. (If, as seems reasonable, we regard the graphitization of cementite as taking place chiefly through a process of solution, diffusion, and re-precipitation, then because diffusion is much more rapid in a molten than in a solid solution, it is but natural that graphitization should be much faster in molten than in solid metal. Prof. Ruff's numbers evidently apply only in the nearly complete absence of silicon, an element which hastens graphitization very greatly. The data in the experiments of Heyn and Bauer[14] indicate that, in their irons containing 4.16, 1.58, and 0.63 per cent. of silicon, the rate of graphitization at temperatures below 1,135° is much greater than that which Prof. Ruff calculates. West's experiments too point to rather rapid graphitization far below the freezing-range in the presence of silicon. See (37)).

(15) *Pro-Eutectic Solidification of Hyper-Eutectic Alloys* (pp. 463, 464).—The cementite in an alloy of carbon content M (Fig. 2) and at temperature M_1 is wholly stable (6), and remains stable till in cooling the alloy starts to cross $B'D'$ at M. The start to cool below $B'D'$ makes so much of the cementite as is in excess of $B'D'$ metastable, and because of this metastability part of it precipitates and at once graphitizes (7). But because the precipitation is not complete, some of the cementite thus metastabilized remains unprecipitated, and hence the molten (as represented by its temperature and dissolved carbon content) passes somewhat to the right of $B'D'$, and follows a path which varies with the rate of cooling, *i.e.*, there is a certain degree of undercooling below $B'D'$. If the removal of heat is rapid, the dissolved carbon content of the molten may fall to BD, as for instance by the path MN. But because on starting to pass below BD, the curve of true solubility, the precipitation is extremely prompt, the molten is not likely actually thus to pass below BD, but rather to slide down along BD

[14] *Stahl und Eisen*, vol. xxvii., pp. 1565 to 1571 and 1621 to 1625 (1907).

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from N to B. In the case of an extremely rapid cooling there may indeed be undercooling even below BD, so that the molten travels along MNB_2. With a moderate (*mittleren*) rate of cooling the molten follows a path like MB_3, *i.e.* with a moderate undercooling below $B'D'$ but not reaching BD. (The line which he actually sketches (Fig. 2) for cooling of moderate rapidity differs slightly from MB_3B, reaching B without previously reaching the line $AB'B$; but his language (p. 464, lines 10 to 12) implies that MB_3 as I give it is that which he actually intended.)

(16) *Near-Eutectic and Eutectic Solidification of Hyper-Eutectic Alloys* (p. 497).—But whether the course of the molten is like MNB_2 or MNB or MB_3, as soon as it reaches the austenite-liquidus AB or its continuation $B'BB_2$ the precipitation of austenite sets in, and the latent heat of this precipitation is added to the still evolving heat of precipitation and graphitization of the cementite metastabilized by the passage to below $B'D'$ (pp. 464 and 497). At the same time the heat supply from the precipitation and graphitization of cementite is greatly increased (p. 497). For the precipitation of the carbon-poor austenite from the carbon-rich molten tends to raise the carbon content of that molten rapidly further to the right of $B'D'$; and the farther it passes to the right of $B'D'$ the greater the metastability and the more rapid the precipitation and graphitization of cementite.

These heat evolutions combined suffice, even if the external conditions are such as to drain heat away relatively fast, to raise the temperature of the molten again to 1,135°, so that it slides up BB' to B', but of course not beyond B', because the start to cross $B'D'$ would arrest the metastabilizing of cementite and the heat evolution resulting from the precipitation and graphitization of that cementite, and without heat from this source a rise of temperature would be impossible.

As the quantity of residual molten decreases, the heat of precipitation from it and the heat of graphitization cease to be able to hold the temperature as high as B'. As the temperature therefore sinks, the course of the molten is naturally along $B'B$, neither falling below $B'B$ because this would further accelerate the precipitation of austenite and thus check the cooling, nor above it for that would arrest completely the precipitation of

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austenite, cut off this important source of heat, and drag the temperature instantly back to $B'B$.

This cooling along $B'B$ continues till the molten reaches B, the true eutectic temperature and dissolved carbon content for the molten. This arrival at SK, this substitution of the true solubility content for mere metastability because in excess of GK, so accelerates the precipitation of cementite (12) that the temperature and dissolved carbon content of the molten now remain constant at B till solidification completes itself eutectic-wise.

The mechanism of the increase in the dissolved carbon content of the molten from GK at B' to SK at B has two features which we may notice. (A) Because the precipitation of the cementite metastabilized by being in excess of GK is incomplete, some of it accumulates in solution in the molten, raising the dissolved carbon content of that molten from GK for the existing temperature towards SK. (B) Because the graphitization of cementite actually precipitated, though rapid, is not instantaneous, some of it (*ein letzter Rest*) accumulates ungraphitized and suspended in the molten, where, acting as bodenkoerper, it in part re-dissolves in the attempt to raise the dissolved carbon content of the molten in the immediate sphere of its influence to SK for the existing temperature. Thus arises a molten enriched beyond GK, and hence normally solidifying at a temperature below B' (p. 497).

Of the eutectic cementite, part diffuses back to enrich towards SK (SE) the early austenite layers deposited when the molten was on $AB'B$ but between B' and B, so that those layers had a dissolved carbon content less than SE; and part graphitizes during the remainder of the eutectic freezing (and thereafter) (p. 497).

Here notice the distinction between the "near-eutectic" co-precipitation of austenite and cementite at points on $B'B$ above B, and the true eutectic precipitation at B. Above B the precipitation of cementite has only such moderate promptness as is due to its metastability because in excess of GK; at B it has the promptness due to the molten's being on the true solubility line, SK.

To sum this up, the course of the molten is $M_1MB_3B'B$.

(17) *Pro-Eutectic Solidification of Hypo-Eutectic Alloys, Under-*

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cooling is Marked.—The approach of the molten part of the hypo-eutectic irons to 1,135° in cooling, occurring theoretically along AB', *i.e.*, without liberation of cementite, they lack the heat evolution due to the prompt and continuous graphitization of cementite which occurs in hyper-eutectic irons as they cool along $B'D'$, towards AB', 1,135°. Lacking this heat supply their undercooling is more pronounced, as, for instance, along the line PQ, Fig. 2 (p. 497).

It is indeed so marked that the temperature at which the molten reaches GK may be as low as 1,125° or even lower (p. 498, top). Till $B'D'$ is reached the solidification is of course of austenite only. On reaching and starting to pass $B'D'$ the cementite in the molten thereby brought into metastability, *i.e.*, excess over GK, starts to precipitate and graphitize. This at first causes only a retardation of the cooling, but as GK is farther exceeded the precipitation and graphitization of the cementite thus made metastable in consequence accelerate. Because of this acceleration of graphitization this retardation of cooling is quickly followed by an arrest of cooling (sojourn at constant temperature), but hardly by a material (*wesentlich*) rise of temperature (pp. 497–98), because the rapidity of graphitization of this metastable dissolved cementite can hardly be rapid enough to cause one, (in view of the presence of the already precipitated austenite, in a position favorable for absorbing its share of all heat evolved).

The further out-crystallizing of austenite enriches the molten in dissolved cementite faster than the lagging precipitation of the metastable cementite can impoverish it, so that the course of the molten is to the right along QQ', and at last it reaches SK at Q'. (As in the case of hyper-eutectic alloys, so it is here. The arrival at Q inducing the precipitation of some cementite, there is "near-eutectic" precipitation in the sense that both austenite and cementite precipitate, though the precipitation of cementite, being only that due to its metastability because in excess over GK, is less prompt than at SK.)

(18) *The Eutectic Solidification of Hypo-Eutectic Alloys* (p. 498).—The arrival at SK, whether at its equilibrium temperature B, or because of undercooling at some point like Q' on its prolongation BB_4, substitutes the prompt precipitation of cementite for the lagging precipitation which has thus far been

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induced by the mere metastability of so much of the dissolved cementite as is in excess of GK. Thus the co-precipitation of austenite and cementite which has been going on now continues, but the acceleration of the precipitation of cementite now induced brings that co-precipitation to the eutectic ratio of austenite to cementite, so that now for the first time there is strict eutectic co-precipitation.

(19) *Recalescence Begins only on the Arrival at SK* (p. 498).—This acceleration of the precipitation and consequent graphitization of cementite so accelerates the heat evolution that now for the first time can a material rise of temperature occur (p. 498, lines 11 to 18). But this rise may not reach the true eutectic temperature, and is the less likely to the greater the initial hypo-eutecticness, *i.e.*, the farther the alloy falls short of the eutectic ratio of 4.3 per cent. of carbon. In short, the molten will now travel along SK from Q' towards or perhaps even to B.

To sum up, the course of the molten in hypo-eutectic alloys is $APQQ'$ and up $Q'B$ to or towards B, Fig. 2.

(20) *Contrast Between Hyper- and Hypo-Eutectic Alloys.*—(A) A recalescence, in the strict sense of an actual rise of temperature, can occur in hyper-eutectic alloys before the dissolved carbon content of the molten reaches SK, *e.g.*, when it reaches $APB'B$ at B_3; but in hypo-eutectic alloys no material rise can occur before the molten thus reaches SK. (B) The undercooling is more marked in hypo- than in hyper-eutectic alloys.

In order to understand this we must distinguish during the solidification between the calorific molten and the inert already solid layers. Whatever heat is generated in the molten must, in order to cause an actual recalescence, be spread over both molten and solid. In the case of hyper-eutectic alloys the quantity of solid, at any given stage in the pre-eutectic solidification, is extremely small, consisting of little more than the small quantity of graphite generated by the decomposition of the cementite thus far precipitated. Thus when a hyper-eutectic alloy in cooling along MB_3 reaches APB there is but an insignificant quantity of solid graphite, and the mass is almost solely calorific molten. Hence the co-precipitation of austenite and cementite which here begins suffices to raise the temperature of the whole, even though the precipitation of the ce-

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mentite is the relatively sluggish one caused by its metastability because in excess of *GK*.

But when a hypo-eutectic alloy in sliding down *APQ* reaches *Q*, there is now present the large quantity of solid carbon-poor austenite precipitated in the cooling along *APQ*. The thermal capacity of this inert solid austenite is so great that the ensuing co-precipitation of austenite and cementite does not suffice to raise the temperature till the precipitation of cementite reaches its full speed on the arrival of the molten at *SK*.

(Two co-operating ways suggest themselves in which the presence of an abundant primary or pro-eutectic substance may increase eutectic undercooling: (A) By mechanically breaking up the residual molten eutectic and isolating its several microscopic particles from each other, it impedes the spread of the influence of the nucleus action of those molecules of the eutectic which are the most prompt to solidify. (B) As for given conditions the undercooling increases, so do the eutectic solidification and the consequent heat evolution hasten. An arrest and a recalescence mean that this acceleration at length suffices first to balance and then to overcome the escape of heat to the environment. Because of the capacity of an abundant pro-eutectic substance, especially if it has great thermal conductivity, both for absorbing heat and for transmitting it to the environment, its presence implies that the arrival at this balance and at this overcoming requires a more rapid heat evolution, and consequently a greater undercooling.

In a word, in a hyper-eutectic alloy the primary solid is extremely scanty because it is almost pure carbon, whereas in a hypo-eutectic one it is extremely abundant because it is carbon-poor austenite. Hence the former may recalesce without the full speed *SK* generation of cementite; the latter cannot).

With the exception of the words enclosed in parentheses the 20 propositions which I have just given are my understanding of Professor Ruff's own interpretation of the course of solidification. Much of it, for instance (8), I do not find distinctly enunciated, but I read it rather hazardously between the lines.

(21) His forecast of the course of the molten during the solidification of hypo-eutectic alloys may be summed up thus: (A) The temperature begins to slacken its fall on reaching *GK*; (B) then stays nearly constant, with perhaps a slight rise;

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till (C) on reaching the prolongation of SK (BB_4) it starts to rise materially; but (D) it may or may not reach the eutectic temperature EB (p. 498).

(22) *General Meaning of Professor Ruff's Diagram*—Looking at this diagram in a general way, we cannot regard it as a "double diagram" in the sense in which these words have often been used. That is to say, it has no lines representing the solubility of graphite in solid and molten iron respectively, or, in other words, the lines at which in cooling the precipitation of graphite begins. $S'E'$ and $B'D'$ do not represent graphite solubility. They represent solely the equilibrium content of cementite in solid and molten iron respectively in the presence of graphite as bodenkoerper, as distinguished from the solubility proper of cementite, *i. e.*, its equilibrium content in the presence of cementite as bodenkoerper. And, because SE and BD represent true solubility, it is the horizontal EB connecting their ends that is the eutectic line. $E'B'$, then, is no true eutectic line, though it is a line at which in equilibrium there is an acceleration of solidification, because on cooling to it both components of the liquid solidify, whereas in the cooling before it is reached, in hyper-eutectic irons it is only their cementite, and in hypo-eutectic ones it is only their austenite, that solidifies. Yet if the true eutectic line is later reached, then solidification is further accelerated. For the precipitation of cementite on reaching its true solubility curve BD is more rapid than the precipitation due to its metastability because in excess of $B'D'$.

So too in undercooling. For instance, though in the case of a hypo-eutectic iron there is a certain acceleration of solidification and of heat evolution when the molten, in sliding down PQ, reaches GK at Q, there is a further acceleration when it reaches the prolongation of SK at Q'. And, in the case of a hyper-eutectic iron, though there is an acceleration of solidification and of heat evolution when the molten undercooled below GK, sliding along MB_3 reaches the austenite liquidus at B_3, there is a farther acceleration when it, in turn, on sliding along AB_3B at last reaches SK at B.

(23) *Ruff on the Temperature of the Eutectic Line EB.*—By applying the foregoing considerations to the cooling curves of Gutowsky, Fig. 4 and Table I., Professor Ruff infers that the

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eutectic temperature is I. below 1,130°; II. above 1,123°; and III. only slightly above 1,123°, so as to be taken reasonably at 1,125°. Proposition II. seems to be well founded; but the gaps in the reasoning by which he infers I. and III. are not readily filled. Let us take these up briefly in the order II., III, and I.

(24) II. *The Eutectic Temperature EB is at or above* 1,123°.—The reasoning by means of which this inference is reached is only hinted at; it may be constructed somewhat as follows:

(A) Gutowsky's alloy No. 4, with most of its arrest at 1,123°, contained 2.184 per cent. of combined carbon. (B) Of this, the largest quantity which the solid austenite could contain in solution is 1.7 per cent., see (11), so that at least 2.184 — 1.7 = 0.484 per cent. must have existed as free cementite during solidification, because, though cementite may be broken up by

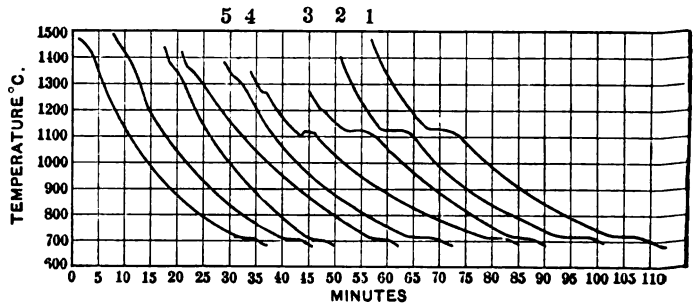

FIG. 4.—GUTOWSKY'S COOLING CURVES.

graphitization, cementite cannot increase in quantity after solidification. (C) The fall of temperature at the end of the arrest was so sharp as to indicate that no important heat evolution occurred after the arrest, and hence this 0.484 per cent. of free cementite must have been generated during the arrest. (D) But the graphitization of free cemetite at a temperature as high as 1,123° is still extremely rapid from Ruff's calculations, say 50 per cent. in 4 sec. Hence the persistence ungraphitized of this 0.484 per cent. of free cementite implies that it must have been generated very near the end of the arrest, and hence very rapidly. (E) But it is only when the carbon content of the molten reaches *SK* (B_4BD, Fig. 2) that free cementite is generated rapidly; hence, when this 0.484 per cent. of free cementite was generated, *i. e.*, during the arrest at 1,123°, the molten

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TABLE I.—*Influence of Total Carbon Content on the Condition of the Carbon, and on the Temperature of the Eutectic Solidification.*

(From the data of Gutowsky, Heyn and Bauer, Carpenter and Keeling, and Wüst.)

Reference No.[a]	Authority.	Silicon.	Carbon Content.						Solidification.		Length of Arrest from Time of Reaching Trough to Again Sinking to Temperature of that Trough.	Rate of Pro-Eutectic Cooling from 1,180° to 1,140°.
			Total.	Graphite.	Combined.	Proportion of Total Carbon Graphitized.	Combined in Excess of 1.3 (E').	Combined in Excess of 1.7 (E).	Trough of Recalescence.	Crest of Recalescence.		
		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	°C.	°C.	Seconds.	Seconds.
1	Gutowsky, First Series.	0.091	4.940	3.910	1.030	79.2	(—0.270)	(—0.670)	1,130	1,138	225	105
2		0.090	4.626	3.413	1.213	73.8	(—0.087)	(—0.487)	1,130	1,134	260	90
3		0.064	3.677	2.263	1.414	61.5	0.114	(—0.286)	1,123	1,130	250	105
4		0.060	2.795	0.611	2.184	21.8	0.884	0.484	1,100	1,123	110	75
5		0.048	2.165	0.282	1.883	13.0	0.583	0.183	1,093	1,093	25	90
6		0.045	1.762	.....	1.762	.....	0.462	0.062	.....	.....	.....	.....
7	Gutowsky, Sec. Series.	.....	3.801	.....	.....	.....	.....	.....	(1,134)	1,134	110	30
8		.....	3.596	.....	.....	.....	.....	.....	(1,130)	1,130	92	40
9		.....	2.742	.....	.....	.....	.....	.....	(1,125)	1,125	52	30
10		.....	1.913	.....	.....	.....	.....	.....	1,062	1,066	13	25
									Temperature of Arrest[b] °C			
11	Heyn and Bauer.[b]	1.58	3.28	2.03	1.25	61.9	(—0.05)	(—0.45)	1,135		270	180 about
12		0.63	3.19	0.25	2.94	78.4	(—1.05)	(—1.45)	1,105		205	180 about
13		4.16	3.12	2.68	0.44	85.7	(—0.86)	(—1.26)	1,133		220	140 about
14	Carpenter and Keeling.	.....	3.98	.....	.....	.....	.....	.....	1,138		240	240 about
15		.....	3.87	2.14	1.73	55.3	0.43	0.03	1,144		.....	
16		.....	3.57	1.38	2.13	39.3	0.83	0.43	1,136		.....	
17		.....	3.42	1.80	1.62	52.6	0.32	(—0.08)	1,114		.....	
18		.....	3.29	0.14	3.15	4.3	1.85	1.45	1,141		.....	
19		.....	3.03	Nil	3.03	Nil	1.73	1.33	1,139		.....	
20		.....	2.12	Nil	2.12	Nil	0.82	0.42	1,110		40	240 about
21	Wüst.[b]	0.012	4.66	3.22	1.44	69.1	(—0.26)	0.14	1,130		270	50
22		0.039	4.04	1.75	2.29	43.3	0.59	0.99	1,122		210	40
23		0.106	3.94	2.28	1.66	57.8	(—0.04)	0.36	1,149		290	50
24		0.013	3.79	2.24	1.55	59.1	(—0.15)	0.25	1,141		350	60
25		0.009	3.76	2.33	1.43	62.0	(—0.27)	0.13	1,138		310	200 about
26		0.015	3.29	0.05	3.24	1.5	1.54	1.94	1,112		130	100 about
27		0.008	3.02	0.37	2.65	12.2	0.95	1.35	1,135		115	80
28		0.006	2.94	0.04	2.95	1.3	1.25	1.65	1,132		80	.....

[a] References: **1** to **10.** Gutowsky, *Metallurgie,* vol. vi., pp. 732–733, and 738 (1909).

11 to **13.** Heyn and Bauer, *Stahl und Eisen,* vol. xxvii., pp. 1566, 1569, and 1570 (1907).

14 to **20.** Carpenter and Keeling, *Journal of the Iron and Steel Institute,* 1904, No. I, pp. 229–231. (The silicon in these alloys was very probably below 0.20 per cent.).

21 to **28.** Wüst, *Metallurgie,* vol. iii., pp. 1–13 (1906).

[b] Nos 11 to 13 and 21 to 28 are based on inverse rate curves. The temperatures here given are those of the greatest retardation. These curves do not show whether there was either an arrest or a recalescence, strictly speaking.

In the Carpenter and Keeling cases Nos. 14 and 20 the cooling curves show no recalescence as distinguished from a retardation. Those for 15 to 19 are not given, and here we are left in doubt as to whether there was a recalescence.

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must have been on or below B_4BD. (F) Moreover, it must have been at or below $AB'B$, because a heat evolution rapid enough to cause an arrest could occur only in case the latent heat of solidification of austenite is then evolving; and austenite can solidify only at or below $AB'B$. In short, from (E) the molten must have been on or below B_4BD, and from (F) it must have been on or below $AB'B$, and hence (G) it must have been on or below B_4BD at or below B. Because its temperature was 1,123°, B must be at or above 1,123°, Q. E. D.

(25) III. *The Eutectic Temperature EB must be only slightly above* 1,123°.—The reasons for this inference are hinted at so vaguely that the gaps cannot be filled with confidence. One conjectures that the reasoning is somewhat as follows. Whereas the difficulty is to explain how so much cementite remains ungraphitized in Gutowsky's No. 4 iron, in spite of the rapidity of graphitization at the high arrest temperature, 1,123°, at which it was generated; and whereas it is only the undissolved or free cementite that is subject to graphitization; and whereas, in so relatively rapid a cooling as Gutowsky's, the cementite precipitated after the arrest and during the subsequent cooling would graphitize but slightly, so that it is the persistence of the cementite free at the time of the arrest at 1,123° that needs explanation; therefore the difficulty is reduced to a minimum by assuming at the time of the arrest the largest quantity of dissolved and the smallest quantity of free cementite, *i.e.*, by assuming that the dissolved cementite represents 1.7 per cent. of carbon, the maximum solubility, or at least that it represents a quantity only very slightly less. If the carbon content represented by the dissolved cementite were 1.7 per cent., then because that is the carbon content of austenite deposited at eutectic temperature, represented by point E on SE, it would follow that the temperature at which it was deposited was the eutectic temperature. If the dissolved cementite were only very slightly below 1.7 per cent., then because the temperature on SE at which such austenite is deposited is only very slightly below the eutectic temperature, it follows that the observed temperature of deposit and arrest, 1,123°, is only very slightly below the eutectic temperature.

(Note the distinction between the carbon content of the austenite deposited and that of the layers which deposit. If we

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were to assume that, in undercooling, the carbon content of the layers which deposit is represented by points on the prolongation of $AE'E$ (Fig. 3), it would still remain true that the dissolved carbon content of the austenite in those layers should as a maximum be represented by SE, for SE represents the maximum solubility of carbon in austenite, the solubility when cementite is bodenkoerper; and if the depositing layers contain more carbon, then that excess should be present as free cementite, or as graphite resulting from its graphitization.)

In short, the difficulty of explaining the persistence ungraphitized of so much cementite would be reduced to an absolute minimum by assuming that the observed arrest temperature, the crest of the recalescence, 1,123°, is the eutectic temperature, and it is reduced nearly to a minimum by assuming that it is only very slightly below the eutectic temperature, say 2° below it.

The argument is rash. For if, as it implies, 0.484 per cent. of carbon in free cementite which existed at the time of the arrest may persist ungraphitized, then the antecedent improbability that a somewhat larger quantity, say 0.584, has so persisted is certainly not great. But if 0.584 has persisted, and if no considerable graphitization occurred after the arrest, then the dissolved carbon of the austenite at the arrest time was $2.184 - 0.584 = 1.6$ per cent. But in that case, if, as we assume, the dissolved carbon of the austenite at the eutectic temperature is 1.7 per cent. or 0.10 per cent. more than the dissolved carbon content of the austenite at the arrest, 1,123°, then the eutectic temperature must be 50° above 1,123°, because the slope of SE calls for a rise of 5° for each 0.01 per cent. increase in dissolved carbon, or, in short, it must be 1,173°. In short the reasoning is rash, because an only slight and permissible change in one of the premises brings us to a result widely different.

It may be that his ground for setting EB at 1,125° is that, granted that $E'B'$ is at 1,135°, and further that the horizontal gap between the liquidus parts $B'D'$ and BD of GK and SK should form a smooth continuation of the gap between the parts $S'E'$ and SE of GK and SK in the solid, then EB must needs be some 10° below $E'B'$ in order to bring these two gaps thus into match. If so, then 1,125° should be taken not as the observed but as the antecedently probable position of

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the eutectic temperature based on the position assigned to $S'E'$, which position itself is none too well established.

Though it is true that the sharpness and extent of the rise of temperature, 23°, and the steadiness of the arrest at the peak temperature are more suggestive in this case, No. 4, than in Nos. 1, 2, and 3, of arrest temperatures below the true eutectic, yet the mere fact of this greater undercooling would in itself tend to sharpen and increase the rise of temperature. The suggestions, then, that the true eutectic temperature is between 1,123° and 1,130° are indeed interesting, but are they more than suggestions?

(26) (I.) 1,130° *must be Above EB.*—From the fact that the combined carbon content of alloy of No. 3, which, after cooling to 1,123°, recalesced to 1,130, is less than E, viz.: 1.414 per cent. instead of 1.7, Prof. Ruff infers that 1,130° must be above the eutectic temperature, referring the presence of 0.114 per cent. in excess of E' (1.414 — 1.30 = 0.114) to the carbon content of the molten reaching SK towards the end of solidification.

(26A) Serious gaps appear in the reasoning. For all that I now see, the phenomena are explicable as easily on the hypothesis that 1,130°, the recalescence crest of alloy 3, is below the eutectic temperature EB, as on Prof. Ruff's hypothesis that it is above EB. Let us run through the course of solidification, first on the assumption that 1,130° is above EB and then on the assumption that it is below EB. On the former assumption, the course of the molten is somewhat as shown by $P'Q'B$, Fig. 5. Assume undercooling of the austenite (see 18) so that the temperature and carbon content of the molten slide down $P'Q'$ till they reach $Q'B'D'$ at Q', 1,123°. From this point on the heat due to the precipitation and graphitization of the cementite made metastable by crossing to below $Q'B'D'$, added to the latent heat of the simultaneous precipitation of austenite which has been occurring from P' to Q', and still continues, suffices first to retard and then to arrest the fall of temperature, all as shown in Gutowsky's curve No. 3, and then to cause a rise to 1,130°. The precipitation of austenite continues to enrich the molten in cementite faster than the metastability of this excess of cementite leads to its precipitation and graphitization, with the result that the molten increases in

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dissolved cementite content, and towards the end reaches *B*. Thus the temperature and carbon content of the molten follow a path like *Q'B*, solidification naturally accelerating and completing itself on reaching *B*.

The arrival of the molten at *B* causes it to deposit layers of carbon content *E*, 1.7 per cent.; but yet the fact that the average combined carbon content of the whole, 1.414 per cent., is far less than *E*, is, indeed, only slightly above *E'*, 1.3 per cent., is very easily understood. For (A) the earliest deposited layers, deposited before selective freezing had enriched the molten

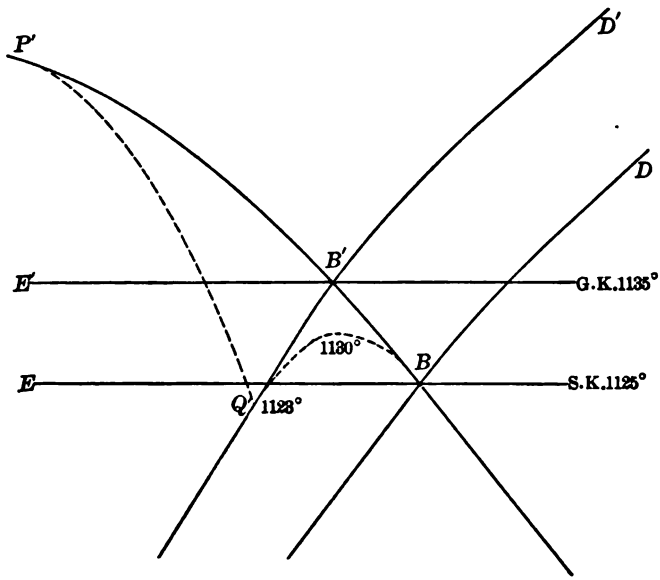

FIG. 5.—THE COURSE OF THE MOLTEN DURING THE SOLIDIFICATION OF GUTOWSKY'S CAST-IRON NO. 3, ON THE ASSUMPTION THAT THE EUTECTIC TEMPERATURE IS 1,125°. (24.)

materially, were, of course, very poor in carbon; and diffusion has had insufficient time to enrich them very greatly; and (B) all the intermediate layers deposited during the progressive enrichment of the molten up to *B* have been deposited with a dissolved carbon content less than *E*, 1.7 per cent., and these, too, have been only incompletely enriched towards *E* by diffusion during the limited time.

(27) *The Eutectic Temperature may be Above* 1,130°.—Next assuming that the eutectic temperature is above 1,130°, say at

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1,135°, let us represent the course of the molten of alloy No. 3 with a very different temperature scale, as in Fig. 6, assuming the same temperature-course as found by Gutowsky and shown in Fig. 5. With this temperature assigned to *EB*, the observed course of the molten recorded by Gutowsky, Fig. 4, agrees with propositions 12, 17, and 18, as we shall now see. Admitting the undercooling called for by proposition 17, the molten crosses

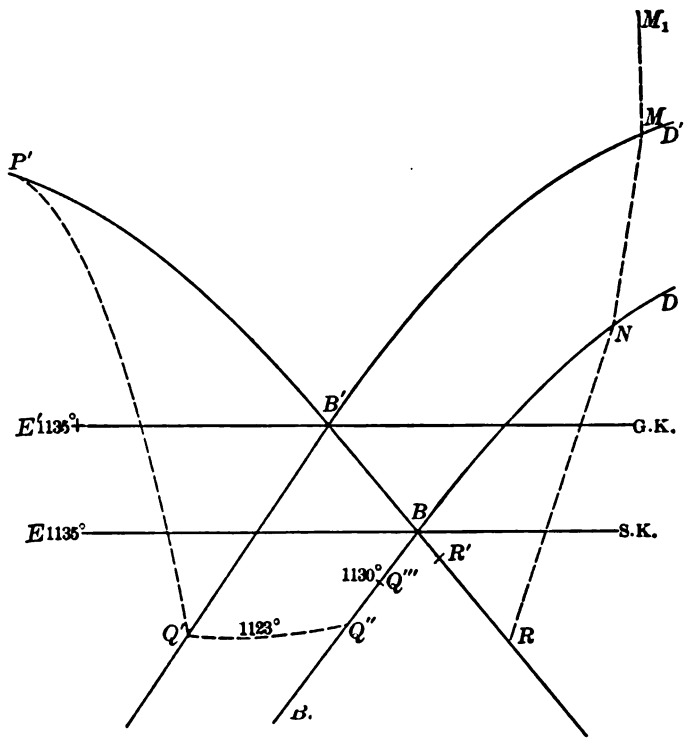

FIG. 6.—THE COURSE OF THE MOLTEN DURING THE SOLIDIFICATION OF GUTOWSKY'S CAST-IRON NO. 3, ON THE ASSUMPTION THAT THE EUTECTIC TEMPERATURE IS 1,135°. (25.)

GK (*D'B'Q'*) at *Q'*, when the metastabilizing, precipitation, and graphitization of cementite suffice first to retard and then to arrest the cooling at the observed temperature of 1,123°. Because, as before, the precipitation of carbon-poor austenite enriches the molten in carbon faster than this metastabilizing and precipitation of cementite can rob it of carbon, the dissolved carbon content of the molten progressively increases

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from Q' towards Q'', and with it increases the carbon content of the austenite of the layers precipitating from it, till the molten reaches SK (B_4BD) by a path like $Q'Q''$.

The acceleration of the precipitation and graphitization of cementite which now occur causes a recalesence towards B, but not fully to B, stopping at the observed temperature 1,130°, Q'''. The austenite of the layers now depositing has reached SE, with a carbon content falling short of 1.7, E, by only the very small amount representing the assumed temperature gap between Q''' and the eutectic temperature EB, but for the same reasons as on the previous hypothesis the average combined carbon content of the austenite as a whole would naturally be only slightly in excess of E'.

Indeed, proposition (19) seems to imply that the eutectic temperature must be at least as high as the observed crest of this recalescence, 1,130°, because, if I understand Professor Ruff aright, in the case of a hypo-eutectic alloy such as this present No. 3, the recalescence cannot, in his opinion, rise above the eutectic temperature. Certainly it cannot raise the molten above APB, Fig. 2; and apparently with equal certainty it cannot raise it above SK, because this would cause a retardation of the precipitation and graphitization of cementite sufficient to check any incipient rise. And if the molten when at 1,130° is neither above APB nor above B_4BD (SK), it cannot be above B, and hence B cannot be below 1,130°. Indeed the curve QB, which I have been obliged to draw in order to make the observed cooling curve agree with his theory that EB is below 1,130°, contradicts his theory set forth in (17) that a material recalescence can hardly occur before the concentration of the molten has reached SK. One is tempted to ask whether Prof. Ruff has not overlooked the fact that the recalescence of this alloy No. 3 rises to 1,130°, a fact shown clearly in Gutowsky's Fig. 617, though not referred to in his table. Section 36 gives evidence tending to set the eutectic temperature much higher, even as high as 1,149°.

(28) *A Possible Simplification.*—For the purpose of holding that the width of the gap between GK and SK in the molten is a smooth continuation of that same width in the solid, it may be important to assume that the true eutectic temperature, EB, is 10° below the "near-eutectic" temperature, $E'B'$, and that

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this is at 1,135°. Yet the phenomena immediately before us seem open to the interpretation that the true eutectic temperature is at least as high as 1,135°, and that the troughs of the recalescences of alloys 1 to 5 represent undercooling below that eutectic temperature. Professor Ruff's extremely ingenious theory may be true, with its "near-eutectic" and true eutectic temperatures, and its view that the undercooling in hyper-eutectic alloys is to between these two temperatures, but in hypo-eutectic alloys to below the lower of them; but if the simpler theory that the governing temperature is the true eutectic temperature, that such undercoolings as occur are undercoolings below it, and such recalescences as occur are attempts to reach it,—if this theory suffices to explain the phenomena at hand, its simplicity would seem to offer a certain advantage, at least in view of the existing conflicts of data concerning the hyper-eutectic liquidus.

Therefore let me run through the phenomena on this simpler theory. On it, and yet adopting Professor Ruff's propositions so far as they do not conflict with simplification, the conditions would be somewhat as follows:

The true eutectic temperature is at least as high as 1,135°. (See 36.)

(29) All that we can infer as to B' from the curve $B'D'$ of Ruff and Goecke, and from their experiments tending to show that BD is not at the left of $B'D'$, is that B' is probably not far above B.

In the hypo-eutectic irons solidification begins with the precipitation of primary austenite, say along $P'Q'$ Fig. 6, followed on crossing the undercooling prolongation of $B'D'$ at Q' by the partial precipitation of that part of the cementite thereby made metastable, which graphitizes more or less rapidly according to the temperature. But because the precipitation of carbon-poor austenite enriches the molten in cementite faster than the graphitization of the cementite metastabilized by the passage to the right of $Q'B'$ impoverishes it, the carbon content of the molten moves to the right, say along $Q'Q''$ till SK, BB_4, is reached, say at Q''. The starting of the carbon content of the molten to cross SK (B_4B) accelerates the precipitation of cementite, the graphitization of which goes on with a rapidity proportional to the existing temperature, and this acceleration

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of precipitation and graphitization of cementite, added to the continuing solidification of austenite, as both now precipitate eutectically, brings the heat evolution to a rapidity which suffices to cause a recalescence, say to Q''', all as already set forth in section (25).

But while the progressive enrichment of the molten in cementite is progressively raising the dissolved carbon content of the austenite of the layers which that molten deposits from $S'E'$ towards and finally to SE, yet in the layers already solid the opposite is continually occurring. In all those austenite layers deposited after the molten has crossed to the right of GK ($Q'B'D'$) some graphite is present, born of the graphitization of the cementite metastabilized by that crossing.

In any given layer, after the deposition of the next succeeding layers has isolated it from the influence of the molten, the graphite thus present is the bodenkoerper. It is indeed true that any free cementite which has precipitated because metastabilized by the crossing to the right of $Q'B'D'$, and has thus far resisted graphitization, would within the sphere of its influence compete with this graphite in the attempt to act as bodenkoerper. But this is a waning influence, because this cementite is continually graphitizing with a rapidity dependent both on the existing temperature and on the quantity of surface which the cementite and the co-existing graphite offer to the solid austenite in which they are bathed. For the solubility of carbon in contact with graphite particles being less than that in contact with cementite particles, the graphite is continously sucking or precipitating out of this austenite all near-by carbon in excess of $S'E'$, and the cementite is simultaneously dissolving in that same austenite in the Danaidic attempt to keep the carbon content of the environing austenite up to SE. The rapidity of this transfer of the carbon of the cementite through solution in the austenite to the microscopic graphite islets will manifestly depend not alone on the temperature, but also on the extent of surface which graphite and free cementite offer to the austenite bath in which they co-exist, and the surface-extent will vary with the capriciously varying form and distribution of the individual islets of graphite and of cementite. Hence great capriciousness of speed of graphitization is to be expected, and actually exists.

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(30) This is the condition of affairs to which I would direct attention, (see (34)), (A) a progressive increase in the dissolved carbon content of the molten, and hence from layer to layer of the depositing solid, with (B) a simultaneous graphitization of the carbon in excess of $S'E'$ in the solid isolated from the molten by later deposited layers, and under the ever growing influence of the solid graphite. So too with any ungraphitized free cementite of the eutectic. The graphitization is rapid at 1,135°; and though it is retarded greatly and capriciously with fall of temperature, yet it continues at far lower temperatures. To this point I will return. (See 37.)

(31) Turning now to the Gutowsky cases in the concrete, Table I. and Fig. 4, note first the progressive lowering of both trough and crest of the recalescence from 1,130° and 1,138° in No. 1 to 1,093° in No. 5, with progressive fall in the total carbon content. This descent of the recalescence with decreasing carbon content is parallel to the familiar descent in the case of Ar1, and may be referred to like causes, especially to the greater undercooling effect of the larger mass of pro-eutectic austenite, wholly irresponsive to the passage past EB.[15] Next note the progressive decrease in the proportion of the total carbon which graphitizes as the carbon content decreases. Thus 79 per cent. of the total carbon graphitizes in No. 1, and only 21 per cent. remains combined, whereas in No. 5 only 13 per cent. graphitizes, and in No. 6 none does. This seems readily explicable by two very simple influences. 1st, the progressive lowering of the temperature of solidification and the great retardation of the graphitization consequent on this lowering; and 2d, the progressive enfeeblement of the graphitization at all lower temperatures through this restriction of the quantity of graphite formed at temperatures at which graphitization is rapid. Graphitization is a process which should accelerate itself. The graphite is a sponge which sucks the dissolved carbon out of the solid austenite, and thus leads the cementite present to dissolve, so that the more graphite is present the faster should be the graphitization of the residual cementite.[16] Witness Gutowsky's finding that, though in the

[15] See also Howe, *Proceedings of the American Society for Testing Materials*, vol. xi., p. 303 (1911).

[16] *Metallurgie*, vol. vi., p. 739 (1909).

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initial absence of graphite the graphitization of cementite was nil during 3 hr., at 1,080° to 1,090°, quite as in Tiemann's [17] experiments, yet a mere prolongation of exposure to this same temperature caused complete graphitization of the whole of the free cementite in 24 hours.

(32) The course of solidification of a hypo-eutectic alloy has just been shown in (29) to agree with this general conception. Turning now to the course of the molten of a hyper-eutectic alloy, it may be conceived as along M_1MNR in Fig. 6. That is to say the molten of carbon content M and originally starting at temperature MM_1 cools without solidification till, on crossing $B'D'$ at M, the cementite thereby metastabilized begins precipitating, and, because of the very high temperature, as it precipitates graphitizes very rapidly. This outfalling of cementite moves the carbon content of the molten proportionally to the left. The undercooling below BD, which Prof. Ruff confines to cases of very rapid cooling, I assume to occur even with Gutowsky's rate of cooling, an assumption which seems to me perfectly admissible. The crossing of BD at N accelerates the precipitation and graphitization of cementite, and the carbon content of the molten shifts more rapidly to the left, striking $P'B'B$ at its prolongation R. The arrival at this line causes the co-precipitation eutecticwise of austenite with the cementite still precipitating, and this addition to the heat evolution suffices to cause a recalescence along RB, say to R'. Thus may be explained the recalescence which occurs in both of Gutowsky's hyper-eutectic alloys.

(33) The variations in the combined carbon content of Gutowsky's alloys explain themselves readily as representing (A) the fact, brought out in column 7 of Table I., that, as the total carbon content decreases, the proportion of that total carbon which graphitizes decreases; and this in turn readily explains itself through (B) the fact that the smaller the total carbon content, the greater is the undercooling, and consequently (C) the lower the temperature at which the cementite, initially dissolved in complete stability, precipitates and thus becomes graphitizable; (D) that the lower this temperature the slower

[17] *The Metallographist*, vol. iv., p. 319 (1901). Tiemann's iron contained 0.255 per cent. of graphite initially, which did not increase appreciably during the heatings.

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is graphitization; and (E) that the more graphite is present, the faster is graphitization. It is necessary to bear in mind both that cementite can graphitize after solidification has completed itself, indeed at much lower temperatures; and that this graphitization is capricious, both of which points I touch on elsewhere. (See 36 and 37.)

(34) Thus there is a race between two opposing tendencies, (A) the progressive enrichment of the molten, and the consequent enrichment of the austenite layers depositing, tending to increase the final resultant combined carbon content, and (B) the tendency to lower that combined carbon content through graphitization. The enrichment of the molten we have seen in §§ 26 and 27 tends to raise the dissolved carbon content of the successively deposited layers up to SE, and on reaching SE to lead to the birth of eutectic throughout the remainder of solidification, an influence tending to raise the combined carbon of the whole. On the other hand graphitization tends to lower that combined carbon content to $S'E'$ for the existing temperature, 1st by graphitizing all free cementite and thus eliminating it as a bodenkoerper, and 2d by precipitating and graphitizing all dissolved cementite in excess of $S'E'$ for the existing temperature.

Thus the various Gutowsky data explain themselves readily on this simplified hypothesis that the true eutectic temperature is as high as 1,135°, and that all the undercoolings are undercoolings below the true eutectic temperature, if we admit that with such a rate of cooling as Gutowsky's undercooling may pass below BD. The question then is, "Is this admission justifiable?"

(35) As regards the effect of a rapid cooling, note that on one hand it tends to lessen the quantity of combined carbon by restricting the diffusion of the carbon back from the later deposited and richer austenite layers and from the eutectic austenite and cementite, if any, to the earlier deposited and poorer layers, so that at the end the austenite on an average of its whole will contain less dissolved and hence perfectly stable ungraphitizable cementite, and more of the cementite will be present, either metastable at the right of $Q'B'D'$, and thus precipitable and graphitizable, or even in the molten on B_1BD, and hence quickly precipitating and thereby graphitizable. But on the

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other hand rapidity of cooling lessens the opportunity for graphitization of the cementite made free, both by shortening the time available and by leading to a lower birth temperature. Thus by increasing the undercooling of the austenite it lowers the temperature at which $Q'B'D$ and B_4BD respectively are crossed, *e. g.*, from B' and B to Q' and Q'', *i. e.*, the temperature at which the precipitation of cementite begins and that at which it accelerates critically.

Experience proves abundantly that, under all usual conditions, this graphitization-impeding effect of rapidity of cooling outweighs its contrary effect in restricting diffusion.

(36) *Capriciousness of Graphitization, i. e., Important Influence of Unnoticed Variations in the Conditions.*—From the fact that the graphitization of cementite should occur, at least in large part, through the action of the graphite nuclei in sucking the carbon out of the austenite, and thereby leading to the dissolving of the free cementite, that this sucking action occurs at the surface of the particles of graphite, and that the dissolving of the cementite occurs at its own surface, and, further, that the extent of surface of each is likely to vary capriciously with the exact conditions of crystallization, we naturally expect the rate of graphitization to be capricious. Our general knowledge of the subject agrees with this expectation. But specific instances are at hand. Thus, Ruer and Iljin,[18] experimenting on two apparently like pieces of gray cast iron, initially with 0.50 per cent. of combined carbon, Nos. 1 and 5 of Table II., heated both to 1,100°, held them there for 3 hr., and then cooled them during 3 hr. to 800°. Thus far all the conditions seem to have been identical. At 800° the dissolved carbon content should, according to Ruer and Iljin's line, have fallen to about 0.75 per cent., and the rest of the combined carbon naturally should have existed as free cementite. Arriving thus at 800°, No. 5 was cooled pretty rapidly, in from 10 to 15 min., to room temperature; No. 1 was held at 800° for 12 hr. and then quenched. The one held 12 hr. at 800° had 1.27 per cent. of combined carbon, practically that which it should have held in solution at 1,100°, so that there had been no graphitization either during the 3 hr. cooling to 800° or during the 12-hr. sojourn at

[18] *Metallurgie*, vol. viii., p. 100 (1911).

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TABLE II.—*Capriciousness of Graphitization in the Experiments of Ruer and Iljin.*

No. 1.	No. 5.	No. 7.	No. 8.
1,100° for 3 hours.		1,100° for 3 hours.	
3 hours.		Quenched.	
800°		Room T°	
800° for 12 hours.	800°	900°	
Quenched.	10 to 15 min.	3 hours.	
Room T°	Room T°	800 for 4 hours.	800 for ½ hour.
		2 hours.	
		700° for 1 hour.	
		Quenched.	
		Room T°.	
Combined Carbon. Per Cent. 1.27	Combined Carbon. Per Cent. 0.73	Combined Carbon. Per Cent. 0.93	Combined Carbon. Per Cent. 0.57

800°. The other, cooled somewhat quickly immediately on reaching 800°, contained only 0.75 per cent. of carbon, or that corresponding to the solubility of carbon at 800°, on Ruer and Iljin's line. Thus in the former case during 3 hr. cooling plus 12 hr. sojourn at 800° no graphitization occurred, whereas in the latter case complete graphitization of the carbon in excess of $S'E'$ occurred during the 3 hr. cooling, without any sojourn at all; or, in short, the graphitization was most capricious.

It is true that they try to explain away the discrepancy between their Nos. 1 and 5 by imagining that the 0.54 per cent. extra graphitization of the combined carbon of No. 5 occurred during the 10 to 15 min. slow cooling from 800° to the room-temperature, explaining that while graphitization does not take place between 1,100° and 800°, it does below 800° if there has been a sojourn above 800°. But this explanation should be accepted very cautiously, (A) because the graphitization of as much as 0.54 per cent. of combined carbon in the almost complete absence of silicon in a 15-min. cooling from 800° to the room-temperature is far more rapid than our general knowledge of the conditions of graphitization leads us to expect; (B) because this explanation fails completely to explain the very great differences between their Nos. 7 and 8, which apparently had equiv-

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alent sojourns above 800° and identical conditions below 800°; and (C) because in my pure cast-iron described in this section, though there was long exposure to temperatures between 800° and 1,000°, graphitization on subsequent very protracted exposure to temperatures below 800° was extremely slow.

Again, in their cases 7 and 8, the conditions were almost identical, as shown by the parallel columns in Table II., yet the ungraphitized carbon was 63 per cent. greater in No. 7 than in No. 8.

Again, a specimen of very pure cast-iron, prepared under my directions by my assistant, A. G. Levy, by melting Professor Burgess's pure electrolytic iron under an excess of sugar charcoal, certainly seems to have retained its combined carbon much more tenaciously than the irons of Ruer and Iljin, having no less than 1.14 per cent. of combined carbon out of a total carbon content of 3.494, after prolonged exposure to temperatures both above and below 800°. The treatment of this specimen for the purpose of graphitizing the cementite consisted first of a 2-hr. cooling from 1,100° to 900°–1,000°, followed by a moderately slow cooling to the room-temperature, then of five slow coolings from 815° and from 790°, aggregating at least 11 hr., in order to graphitize the pro-eutectoid cementite precipitating at these temperatures, and finally in very long holdings, 79.5 hr. between 635° and 710°, and 118.5 hr. between 700° and 750°, for the purpose of graphitizing the cementite formed on cooling below the "near-eutectoid" temperature, *i. e.*, the junction of *S'E'* and *GOS*. Though at the end of the 79.5 hr. holdings the combined carbon was reduced to 1.14 per cent., yet some prominent crystals of cementite persisted ungraphitized even till the end of the additional 118.5 hr., as shown in Fig. 8. They were shown to be cementite by darkening with sodium picrate. The persistence of the initial ingot structure is shown by the well-marked rectangular areas in Fig. 9 of this same specimen after the six slow coolings. The white dendrites are evidently the relics of the layers first to solidify, and therefore poorer in carbon than those which solidify later. The persistence of this dendritic structure shows how slow diffusion may be. Other instances of very tardy graphitization are given in Appendix I.

In view of this great capriciousness of the rate of graphitiza-

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tion, the quantitative determinations of this rate by Ruff, based on these few cases of Gutowsky's, should not, I believe, receive great weight, especially because the variations in the silicon of the Gutowsky irons were such as to contribute to the decrease of graphitization with decreasing carbon content. Thus his hyper-eutectic irons had 0.09 per cent. of silicon, whereas his hypo-eutectic ones had only from 0.045 to 0.064. Again though the slower cooling of Carpenter and Keeling's alloy No. 20 of Table I. should have favored graphitization, it had no graphite, though the closely corresponding No. 5 of Gutowsky had 0.282 per cent.

Again the combined carbon content in Carpenter and Keeling's irons cooled under apparently like conditions varied capriciously, as shown by Nos. 14 to 20 in Table I. Though there is indeed a general decrease in graphitization as the carbon content falls, yet the irregularities in this decrease are greater than we should expect from unrecorded variations in the conditions.

Like variations occur among the Wüst cases in Table I., but these seem referable to the capricious variations in the temperatures at which the arrests occur, which variations themselves are unexplained as noted below.

Further indications of considerable capriciousness are found (1) in the much higher arrest or recalescence temperatures of some of Carpenter and Keeling's alloys, *e.g.*, Nos. 15, 18, and 19, than in the corresponding ones of Gutowsky; in the irregularity with which this arrest temperature varies with decreasing carbon content in the Carpenter and Keeling alloys; and in the imperfect accord between Gutowsky's first and second series. The more rapid cooling in his second series would be expected to favor undercooling; yet the undercooling seems on the whole less than in his first series, for like carbon content.

The Wüst cases in Table I. give like evidence. First their arrest temperature even in hypo-eutectic irons rises to 1,149°, or far above any corresponding temperature reached under like conditions in the Gutowsky data, and reminding us that Wittorff puts the "near-eutectic" temperature as high as 1,180°. Second they vary capriciously among themselves, the two lowest in carbon, Nos. 27 and 28 with 3.02 and 2.94 per cent. of carbon respectively, have their arrests at 1,135° and 1,132°,

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or well above the arrests of alloys much richer in carbon, *e.g.* Nos. 22 and 26, with 4.04 and 3.29 per cent. of carbon respectively, and with arrests at 1,122° and 1,112°.

The alloy richest but one in carbon, No. 22, has its arrest at a temperature much below that of any of the others except No. 26. The only hyper-eutectic alloy, No. 21, has its arrest temperature lower than the alloy poorest in carbon, No. 28, and lower than four other hypo-eutectic alloys, Nos. 23, 24, 25, and 27. In some of these cases the variations in the rate of cooling go towards explaining the discrepancies, but not the discrepancy between Nos. 22 and 23. Note that though Nos. 24 and 25 have almost identical carbon content, and though the much slower cooling of 25 should tend to raise its arrest temperature, in fact the temperature is 3° lower than in No. 24.

The interpretation of this evidence should be extremely cautious. The most striking thing about it is that, whereas each of the Gutowsky series is concordant taken in and by itself, with the arrest or recalescence temperature as the case may be falling progressively with the carbon content; and whereas the anomalies in the Heyn and Bauer cases may readily be due to the variations in silicon content; yet the Carpenter and Keeling and the Wüst results are not concordant *inter se*, the arrest temperature having little relation to the carbon content. Even if we were to challenge the Carpenter and Keeling results because the cooling curves themselves are not given, the Wüst difficulty would remain, for these cases rest on legible inverse-rate curves, with very sharp retardations.

In view of the antecedent probability that the eutectic temperature should fall with decreasing carbon content; of the continuous fall in each of the Gutowsky series; of the apparent very great care with which he worked; and of the possibility that these earlier investigators may have overlooked some needed precaution; one comes away with the impression that the law is probably true, but that the unnoticed variations in the conditions may easily obscure it under the intrinsically difficult observational conditions. We can at least say this: that in each series in which there is a carbon content less than 2.74 per cent., this further lowering of the carbon content is accompanied by a marked lowering of the arrest temperature. Further evidence is awaited.

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(37) *Graphitization Continues at Temperatures far below the Freezing-Range.*—If we accept the line of Ruer and Iljin, then it is clear that, in Gutowsky's alloy No. 1, graphitization continued at least down to 1,000°, for in no other way can we explain how its combined carbon content comes to be so low, 1.03 per cent. For if as long as the cementite is dissolved in the austenite in quantity not to exceed $S'E'$ it is perfectly stable and ungraphitizable, then graphitization could not reduce the combined carbon content to 1.03 per cent. as it did, till the temperature had sunk to 1,000°, at which the solution and hence ungraphitizable carbon content for the first time falls as low as 1.03, only that in excess of 1.03 being graphitizable.

Again in the malleablizing process the combined carbon content often becomes negligible. In the case given in Table III. it is only 0.022 per cent. Benedicks[19] too graphitized completely the pearlitic cementite of a gray-iron by seesawing at temperatures which rose at the highest to but little above 800°. Further I have collected three trustworthy cases in which commercial cast-iron contained no combined carbon, together with one in which it had only 0.04 per cent.[20]

But the ungraphitizable dissolved carbon content evidently cannot fall so low at any point on $S'E'$ above S', the GK or "near-eutectoid" temperature, $A'1$, at which $S'E'$ cuts GOS at carbon content about 0.70 per cent. On cooling past $A'1$ a transformation parallel to that which occurs in cooling past $A1$ should occur; *i.e.*, the austenite under the domination of graphite as bodenkoerper should here break up into a conglomerate of ferrite and graphite, just as, when cementite is bodenkoerper, austenite breaks up at $Ar1$ into ferrite and pearlitic cementite. Hence when we find the combined carbon less than the $A'1$ carbon content, say 0.70 per cent., we infer that graphitization has continued to below $A'1$.

Ruer and Iljin's line implies that $A'1$ is at least as low as 775° even if we accept the high position assigned to GOS by Goerens and Meyer.[21] This then implies that, at least in the presence of silicon as in the manufacture of malleable cast-iron, graphitization continues habitually at least down to 775°.

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[19] *Metallurgie*, vol. v., p. 42 (1908).

[20] *Trans.*, xxxix., 58 (1908).

[21] Goerens and Meyer, *Metallurgie*, vol. vii., pp. 307–312 (1910).

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Ruer and Iljin's paper has confirmatory internal evidence (their Table I.) tending to show that *A'1*, the near-eutectoid temperature, is between 700° and 800°; for on reheating graphitic cast-irons in which the combined carbon content had been brought very low, there was marked increase in the combined carbon when the reheating temperature was 800° but none when it was 700°. At *A'1* re-combination up to *S'E'*, 0.7 per cent. of carbon should occur, the graphite present dissolving in the austenite to that extent, but we hardly look for such recombination below *A'1*. It may indeed be true that external graphite will dissolve in iron between *A1* and *A'1*, the small quantity of pearlitic cementite present dissolving to form a small quantity of austenite on rising past *Ac1*, and the external graphite cementing that austenite. But internal graphite present so abundantly as in these irons of Ruer and Iljin would be expected to act so efficiently as bodenkoerper that, at all temperatures below *A'1*, the solubility of carbon in the presence of graphite being nil here, it would precipitate all dissolved carbon as fast as it dissolved. Further evidence on this point is needed.

But even in Ruer and Iljin's alloys nearly free from silicon (0.05 per cent. of silicon with 0.21 per cent. of manganese) the combined carbon content fell to 0.27 per cent., which by the reasoning just set forth, shows that graphitization must have continued at least down to *A'1*, *i. e.* at least down to 775°.

And whatever reasonable position we assign to *GOS* and *S'E'*, the frequent low combined carbon content of very graphitic malleable castings, the occasional like combination in gray cast-iron, the low combined carbon of Ruer and Iljin, and the complete graphitization by Benedicks, show with mathematical certainty that graphitization continues down to temperatures very far below the freezing range unless our present theories are far wrong.

That graphitization goes on with very considerable rapidity in the presence of silicon at temperatures below the freezing range is shown by evidence from several courses.

Heyn and Bauer[22] and Goerens and Gutowsky[23] show this by determining the degree of graphitization which occurred at various stages of cooling above, in, and below the solidification

[22] *Stahl und Eisen*, vol. xxvii., pp. 1565-1571 and 1621-1625 (1907).

[23] *Metallurgie*, vol. v., p. 145 (1908).

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range. This they did by allowing a series of specimens to cool slowly to a series of known temperatures in and near that range, and then interrupting the cooling by quenching in water, so as to arrest the graphitization at the stage which it had reached at each temperature. Chemical analysis of the quenched specimens showed in every series that graphitization had gone on to an important degree below the solidification range.

This is further indicated by the reported industrial observation that "castings made in chill moulds show—no signs of chilled structure, provided the castings are removed immediately after casting."[24] Had they been left in contact with the chill they would have remained chilled. Because the conditions up to the moment of removal are identical whether the casting is left in contact with the chill or is removed, and because if left in contact with the chill its face is chilled, *i. e.* graphiteless, it follows that at the moment of removal the face of removed castings is graphiteless. But because it is solid when removed from the mold, and because after the ensuing relatively rapid cooling it is found to be gray, it follows that graphitization in these cases is habitually rapid below the freezing-range.

T. D. West[25] found that to reheat chilled cast-iron plates to temperatures well below the melting-point by pouring molten cast-iron against them caused slight graphitization, as inferred from the appearance of the fracture. With further reheating the graphitization increased, till when the temperature of the face reached the melting-point, the face of the plate had been brought about one-quarter way towards the condition of gray iron. In each case the reheating was rapid and brief.

He further finds that if a cast-iron casting, after the outer part has solidified and cooled at a normal rate to somewhat below the eutectic freezing-point and thus become graphitic, is suddenly withdrawn from the mold and quenched in water, the inside is of white iron. The fact that the inside of the casting is white though the normal slow rate of cooling has extended below the eutectic freezing point, shows that it solidified as white iron. The graphitization of the outside must have occurred in that phase of the cooling by which the gray outside had outrun

[24] P. Munnoch, *Iron Age*, vol. lxxxix., p. 1414 (June 6, 1912).
[25] *The Foundry*, vol. xl., pp. 245–251 (June, 1912).

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the white interior at the moment when the quenching occurred. Because the amount by which the cooling of the outside has outrun that of the inside must be very slight, we infer that this graphitization occurred in a very narrow range of cooling and presumably below the eutectic freezing point. His further ingenious experiments show that, if the slow cooling had not been interrupted thus, the interior would have been very gray.

(38) *How Dominant is the Influence of the Bodenkoerper ?*—There is one fact which is not easily reconciled with one of the foregoing conceptions and two of the foregoing assumptions. The conception is (1) that the nature of the bodenkoerper, *e. g.*, whether it is free cementite or free graphite, determines whether the dissolved carbon content of austenite shall be represented by SE or $S'E'$. The assumptions are (2) that SE is truly represented by the determinations of Gutowsky, and (3) that $S'E'$ is truly represented by those of Ruer and Iljin.

The fact difficult to reconcile with the foregoing is that the difference between the conditions in certain of Gutowsky's experiments and the conditions in Ruer and Iljin's seems insufficient to explain why cementite should have dominated completely in the former, and graphite completely in the latter, bringing the combined carbon only to $S'E'$.

In Gutowsky's No. 42 with gray iron and 43 with white iron, in spite of the final presence of over 2 per cent. of internal graphite the dissolved carbon content of the austenite followed the line SE with great precision; whereas in Ruer and Iljin's in the same range of temperature and with about the same quantity of internal graphite it followed the line $S'E'$. The sole material difference in conditions appears to be that Gutowsky's cylinders of cast-iron were imbedded in a carburizing substance, while Ruer and Iljin's were not. They were enclosed in sealed quartz tubes. There were present in these tubes, 1st the negligible quantity of air present at the time of sealing, and 2d whatever gases were evolved by the iron itself during the prolonged heatings.

Taking these results at their face they seem to teach that the influence of external animal charcoal in Gutowsky's Nos. 42 and 43 has completely dominated and indeed effaced the influence of about 2.1 per cent. of internal graphite. But this is so improbable antecedently that we hesitate to accept it. For this

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internal graphite, at least in No. 48, must have been very finely divided. Therefore, distributed as it was throughout the whole mass of the austentite and thus offering enormous and most intimate surface exposure, its efficiency as bodenkoerper certainly ought to be much greater than that of the external animal charcoal simultaneously present, which could at best offer only slight and not intimate contact.

Two escapes from this situation offer themselves. (1) The assumption that the influence of internal graphite, even when finely divided and distributed internally through the iron, in dragging the dissolved carbon content back from SE towards $S'E'$, is so feeble that it is completely negligible when it competes with the influence of external amorphous carbon in cementing the austenite up to SE. This path of escape from the difficulty is certainly a most thorny one.

(2) The influence of the bodenkoerper may be altogether secondary to that of some unnoticed difference between the conditions of Gutowsky and those of Ruer and Iljin, a difference which led the former always to SE and the latter always near to $S'E'$. To throw light on this I purpose repeating both sets of experiments under identical conditions.

In any event, till this matter is cleared up the results of Ruer and Iljin should be received with some caution for two reasons. (A) for each temperature they differ between themselves by from 0.10 to 0.16 per cent., or from about $\frac{1}{3}$ to about $\frac{1}{2}$ of the gap between their $S'E'$ and Gutowsky's SE. (B) Moreover there is reason to fear that the results of the "absorption" method which they followed may be either too high or too low. This method consists in holding graphitic iron at a temperature for which the $S'E'$ carbon content may be assumed safely to be greater than the initially dissolved carbon content, in order to allow it to absorb the internal graphite till it reaches $S'E'$ and thereby saturates itself. The results thus reached may either be (1) at the left of the true $S'E'$ because of insufficient time to complete the absorption to that line; or (2) they may be at the right of the true $S'E'$, because on unnoticed petty falls of temperature, free cementite would precipitate, which on the subsequent unnoticed rise would compete with the internal graphite in carburizing the austenite, so that their joint action would carburize it to a point between true $S'E'$ and true SE.

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Moreover any free cementite not dissolved in carburizing the austenite would be counted in with the combined carbon and be found by Ruer and Iljin. Against (2) it may be urged that though a like precipitation of free cementite might bring the SE determinations of Gutowsky to the right of the true SE, yet if we may judge from the great concordance of his results it did not. And if it did not in his case, why should it in Ruer and Iljin's?

The "precipitation" method is in a better position, because, while its results may be too high, they can hardly be too low; hence the lowest trustworthy results reached by this method should be taken as the nearest experimental approach to the true $S'E'$. This method consists in holding graphitic iron at a temperature at which its initial dissolved carbon content is more than enough to saturate it, *i. e.*, exceeds $S'E'$, in order to allow the excess of dissolved carbon to precipitate out and to graphitize, then cooling rapidly, and determining the combined carbon. The results of this method may indeed be too high (A) because the precipitation of the excess of the dissolved carbon may lag, and (B) because the graphitization of the cementite thus precipitated may lag; but it is not easy to see how they can be too low.

Unfortunately in the absence of silicon this graphitization of cementite is extremely sluggish, as is shown by the results reached with it by Ruer and Iljin and by myself and Levy ((36) (and Appendix I). In view of this the most promising way seems to be to determine $S'E'$ by the precipitation method with two different percentages of silicon, in order to confirm or disprove the current belief that silicon does not shift the position of $S'E'$ but only hastens the arrival at it. If this belief is confirmed, then the position of $S'E'$ thus found would be its true position. If it is disproved, there is still the possibility of finding the true position of $S'E'$ in silicon-less irons by extrapolation.

(39) *Conclusion.*—We have now seen in (36) that the unexplained variations in the cooling curves are very great; that the two series of observations on this point by Gutowsky do not agree; and that the rate of graphitization is very capricious. In (37) we have seen that graphitization continues at temperatures far below those to which Prof. Ruff seems to con-

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fine it. In (38) we have seen difficulty in reconciling the Gutowsky and the Ruer and Iljin lines with the assumed dominance of the bodenkoerper, all of which seem essential premises underlying Prof. Ruff's reasoning. In view of these facts the data as to arrest temperature and graphitization in the single Gutowsky series used by Prof. Ruff seem to me wholly insufficient for a firm foundation on which to base calculations as to the equilibrium position of the eutectic temperature.

The new data offered in this paper have been reached in experiments carried out in part under a grant from the Carnegie Institution of Washington, in the Metallurgical Laboratories of Columbia University. For the analyses we have to thank Messrs. Booth, Garrett & Blair, and for the electrolytic iron Prof. C. F. Burgess.

Appendix I.—A Case of Slow Graphitization Above 800°.

The experiments here described, made by A. G. Levy under my direction, consisted in exposing to certain oscillations of temperature a specimen of malleable cast-iron, No. 1, and four cylinders, Nos. 2, 3, 4, and 5 of gray cast-iron made by cementing and melting Prof. C. F. Burgess's electrolytic iron. These four cylinders were cut out from a single small ingot, called "Original" in Table III. In making this ingot the electrolytic iron was melted with an excess of pure sugar charcoal in a graphite crucible brasqued with sugar charcoal, bound with a little liquid asphalt and vaseline. The crucible was highly heated in a gas-furnace, opened once to make sure that the mass was molten, and immediately closed. After further holding it was then cooled to about 1,100° by lessening the gas supply, and thence to between 800° and 900° in 4 hr. by further successive reductions of the gas supply.

Four cylinders, Nos. 2 to 5, approximately 2 in. long by 0.25 in. in diameter, cut from this ingot were enclosed each in a sealed quartz tube, and reheated to about 950°, when on account of an accident they were all quenched in water. No. 2A (Figs. 10 and 11) represents the condition after this quenching.

Each of these four cylinders was then enclosed in a steel gas-tube closed at each end with a cap. No. 4 was packed with

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FIG. 8.—PERSISTENCE OF CEMENTITE CRYSTALS IN PURE CAST-IRON AFTER PROLONGED HEATING, INCLUDING 118 HOURS BETWEEN 635° AND 750°. MAGNIFIED 85 DIAMETERS.

FIG. 10.—PURE CAST-IRON NO. 2A QUENCHED FROM 950°. CONTACT OF EUTECTIC WITH GRAPHITE. FINE NEEDLES ARE PRO-EUTECTOID CEMENTITE. MAGNIFIED 85 DIAMETERS.

FIG. 9.—PERSISTENCE OF THE ORIGINAL SOLIDIFICATION DENDRITES AFTER SIX SLOW COOLINGS. PURE CAST-IRON ETCHED WITH SODIUM PICRATE. MAGNIFIED 18 DIAMETERS.

FIG. 11.—PURE CAST-IRON NO. 2A QUENCHED FROM 950°. CONTACT OF EUTECTIC WITH GRAPHITE. FINE NEEDLES ARE PRO-EUTECTOID CEMENTITE. MAGNIFIED 85 DIAMETERS.

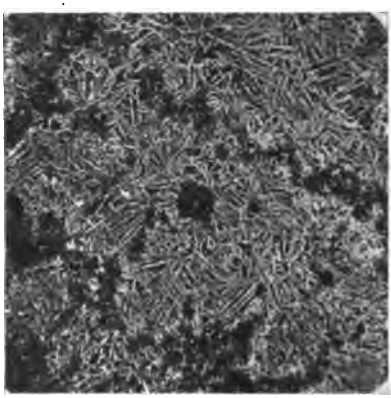

FIG. 12.—PURE CAST-IRON QUENCHED AFTER REPEATED OSCILLATIONS OF TEMPERATURE, FIG. 7. MAGNIFIED 85 DIAMETERS.

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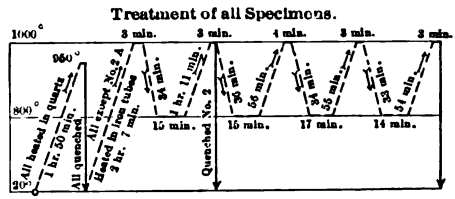
sugar charcoal in its tube, and No. 5 with kish from an iron blast-furnace. The tubes containing Nos. 2 and 3 had no carburizing material, and each was separated from its specimen by means of sheet asbestos.

The specimen of malleable cast-iron, No. 1, was enclosed in a like steel tube.

TABLE III., AND FIG. 7.—*Graphitization and Re-Combination in Certain Cast-Irons.*



	Cast-Iron. Original of Nos. 2 to 5.	Malleable Cast-Iron, Original of No 1.	Cast-Iron No. 2 A. Alone.	Cast-Iron No. 2. Alone.	Cast-Irons. No. 3. Alone.	Cast-Irons. No. 4. Sugar Charcoal.	Cast-Irons. No. 5. Kish.	Malleable Iron No. 1. Alone.
Comb. carbon	1.598	0.022	.....	1.554	1.130	1.542	1.628	1.030
Graphite	1.356	1.952	.....	0.930	0.454	1.040	0.948	0.738
Total carbon	2.954	1.974	.....	2.484	1.584	2.582	2.576	1.768
Silicon	0.040	0.717						
Manganese	None.							
Phosphorus	Not det.							
Sulphur	0.035							

NOTE.—For the chemical analyses we are indebted to Messrs. Booth, Garrett & Blair.

These five specimens, each in its tube, were then heated together to 1,000°, cooled slowly to 800°, and reheated to 1,000°, when No. 2 was drawn and quenched with its containing tube in cold water. The remaining ones were then reheated to 1,000°, cooled slowly to 800°, and thus seesawed up and down as shown in Fig. 7, till on reaching 1,000° for the fifth time they were drawn and quenched in water each in its enclosing tube. The cooling in this quenching was not very rapid, as is shown by the fact that Nos. 2, 3, 4, and 5 could be sawed with a hack-saw, though No. 1, the specimen of malleable cast-iron, could not. No. 3 which had become much decarburized also had decidedly less combined carbon than the original. The procedure and results are shown in Fig. 7 and Table 3.

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Each of the quenched specimens had a very considerable quantity of well oriented cementite needles, part of which may well represent precipitation during the quenching, though part represents the combined carbon present at 1,000° in excess of *SE*, or more likely in excess of some line between *SE* and *S'E'*. The microstructure of Nos. 2, 3, 4, and 5 was almost identical, and is shown in Fig. 12; that of No. 2A, Figs. 10 and 11, had very much fewer fine cementite needles, and instead some rather massive eutectic islands. The massive white islands are in most cases clearly masses of eutectic. Even the arrowhead mass in Fig. 10 is very probably of eutectic. The cementite of this eutectic (Fig. 10) has escaped graphitization both in the original solidification and in reheating. The relative freedom of Nos. 2 to 5 from eutectic suggests that in the successive heatings the cementite of this eutectic has been washed away to an extent hardly to be foreseen, to re-precipitate as fine needles. The immediate contact of graphite with this eutectic in Figs. 10 and 11 is to be noted. Here the acicular regions about the graphite represent the austenite formed at a high temperature along with that graphite from the graphitization of cementite. The needles are the pro-eutectoid cementite which has precipitated out of that austenite in cooling down, as is seen on comparing this structure with Fig. 12.

At the first glance one might take some of these large eutectic masses for cementite, and wonder at the immediate contact of this cementite with graphite, a thing which, after diligent search and inquiry, I have never found save in some micrographs by Prof. Wüst [26] and Prof. Goerens.[27]

The combined carbon content of Nos. 2, 4, and 5 varied only within the limits of 1.542 and 1.628, corresponding roughly to the carbon dissolved between 1,050° and 1,100° according to the line *SE*. Though the total period of exposure to temperatures between 800° and 1,000° was not very great, one certainly would have strongly expected some graphitization of the carbon, reducing the combined carbon towards *SE* for that range, or even below *SE* in view of the large quantity of graphite present. Yet the iron which stayed longest in this range, No. 5, actually had a little more combined carbon than the original. Thus

[26] *Metallurgie*, vol. iii., Figs. 14, 15, 16, opposite p. 12 (1906).

[27] Introduction to *Metallography*, 1903, Figs. 142, 143, and 144, pp. 197–198.

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even in the presence of as much as 1.598 per cent. of graphite, free cementite may persist apparently without undergoing even partial graphitization during a very considerable sojourn rising repeatedly to 1,000°.

The malleable cast-iron is in great contrast to these others. Its combined carbon content rose to 1.03 per cent., which agrees fairly with the maxima of Ruer and Iljin's results. Their highest two combined carbon contents on heating gray cast-iron to 1,000° without carburizing material, and then quenching, were 1.02 and 1.07 per cent.

The extreme slowness of the graphitization of my pure cast-iron is to be referred to its low silicon content.

Appendix II.—Solvent, Solute, and Bodenkoerper.

When we saturate a common aqueous solution, for instance of common salt, by holding it in contact with an excess of solid salt, we call the water the solvent, the salt which is dissolved in the water the solute, and the excess of solid salt needed to insure our saturating the water, the bodenkoerper, or solid phase. When the water has dissolved all the salt which it is capable of containing at this temperature as is shown by its leaving some solid salt unabsorbed we call the solution saturated, and this quantity of salt we call the solubility of salt in water at this temperature.

Admitting that the carbon in molten cast-iron exists as cementite, does the line $B'D'$, Fig. 1, represent the solubility of cementite in molten iron in this same sense? Manifestly not. The quantity of a salt which water will dissolve varies with the nature of the solid supply of that salt, the bodenkoerper, with which that water is in contact.

Roozeboom's[28] classical example of the varying quantity of thorium sulphate which water will contain in equilibrium with different bodenkoerpers is shown in Fig. 13. If water is in contact with the solid salt $Th(SO_4)_2, 9H_2O$ the quantity of $Th(SO_4)_2$ which 100 parts of it can contain at various temperatures is represented by the line ABC. Each of the other lines represents the quantity of $Th(SO_4)_2$ which water can contain when in contact with some one of the other solid hydrated thorium sulphates.

[28] Ostwald, *Lehrbuch der Allgemeinen Chemie*, vol. i., p. 1062 (Leipsic, 1903).

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Water in contact with a solid salt will dissolve that salt till equilibrium is reached, the equilibrium at which the passage of particles of the solute out of the solution to rejoin the bodenkoerper is just rapid enough to balance the simultaneous passage of particles from the bodenkoerper into the solution to join the solute. So two neighboring hot bodies are in equilibrium with each other when they are at the same temperature, *i.e.*, when passage of heat from A to B just balances the passage back from B to A. So water and steam are in equilibrium in

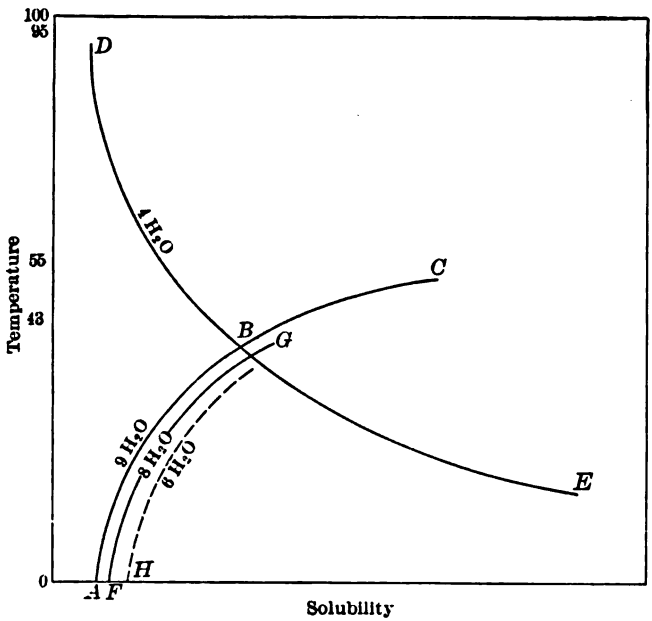

FIG. 13.—SOLUBILITY EXPRESSED AS PARTS OF $Th(SO_4)_2$ PER 100 PARTS OF WATER. THE QUANTITY OF THORIUM SULPHATE WHICH WATER WILL DISSOLVE AT GIVEN TEMPERATURES VARIES WITH THE NATURE (DEGREE OF HYDRATION) OF THE BODENKOERPER OR SUPPLY OF SOLID SULPHATE.

the boiler when the rate of evaporation of the water just balances the simultaneous condensation of steam back upon the surface of that water. This conception of equilibrium as representing not stagnation but equal and opposite transfer is most important.

In the case of a solution we may conveniently say that when this equilibrium is reached the solution is "saturated." But

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the point to recognize clearly here is that the quantity or concentration of the solute in the solution at which this equilibrium is reached, *i.e.*, at which saturation occurs, varies with the nature of the bodenkoerper.

Turning now to the term "solubility," when we speak of the solubility of a salt in water we naturally have in mind the quantity of that salt which the water can contain when it is in equilibrium with an excess of that same salt in the solid state, *i.e.*, as bodenkoerper. We might indeed speak of the "solubility of salt in water when in contact with bodenkoerper A, B, or C;" but to prevent confusion it seems better to confine the term "solubility," (*SK*, *Saettigungskonzentration*) to the case in which solute and bodenkoerper are identical, and to use the term "equilibrium content" (*GK*, *Gleichgewichtskonzentration*) to represent the quantity of salt which the solvent can retain dissolved within it when in contact with any other bodenkoerper.

Philosophically speaking, solubility is only a single case of equilibrium content, representing equilibrium when the solute and the bodenkoerper are identical (apart from ionization), and the distinction between true "solubility, *SK*," and "equilibrium, *GK*" is purely one of convenience.

In the experiments of Ruff and Goecke, and also in those of Hanemann, the molten iron was held in contact not with solid cementite, if indeed that is possible, but with solid graphite. The quantity of carbon which it took up, *B'D'*, therefore was not the true solubility of cementite in molten iron, *SK*, but the equilibrium content *GK* of carbon in molten iron in contact with graphite.

Discussion.

J. E. Johnson, Jr., Ashland, Wis.:—This material is largely beyond the understanding of most of us practical men, but there are certain portions of that diagram that are of enormous importance. We have been conducting a very extensive investigation of the quality of charcoal-iron and of all iron, and we have found that this eutectic point here (indicating on chart) is a sort of balance point between the good irons and the bad ones, and I have some slides to show what practical results we have found depending on this point here on the diagram; and it would be a mistake for men who were concerned with

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operation and were getting practical results, to think that these things are of no importance. They are exceedingly hard to understand, and I confess that a great deal of them is beyond me, but there is no doubt that the quality of iron depends to an enormous extent on the location of the carbon content of the iron to the right and left of this point, and the position of that point is dependent upon the silicon, and I wish to make a plea to the practical and operating members of the Institute not to disregard this material on account of its being highly scientific, because I hope to show later that there are matters of the highest practical importance that depend directly upon certain portions of this diagram.

HENRY D. HIBBARD, Plainfield, N. J.:—I understand that washed metal, which is practically pure iron and carbon, does not solidify at one instant, as cast-iron does, but passes through a mushy stage. That has been an enigma as to how and why it did so, and possibly some light might be thrown on it in this connection.

BRADLEY STOUGHTON, New York, N. Y.:—If Wittorff's propositions are accepted, washed metal containing more than 4.1 per cent. of carbon would begin to solidify as liquid plus Fe_4C along the line *MN*. This line is so very steep that the least bit of carbon above the eutectic ratio would make a very large amount of Fe_4C. So that, almost immediately after the washed metal began to freeze, this much solid separating out would give us a mixture of liquid and solid, and the iron would have to cool from 1,350° down to 1,150°; that is, through a range of 200° C. (which is equal to about 350° F.) before it would be entirely solid; and that would explain this mushy state—part liquid and part solid—through which it would pass.

R. H. SWEETSER, Columbus, Ohio:—Is there any way of applying the information brought out in these charts to the fact that in making very low silicon basic iron there are times when the silicon will be down between 0.2 and 0.3 per cent. and the sulphur under 0.03, when the pig-iron will be gray, with a little "bloom" of carbon deposited on the pigs just as soon as they chill, and at other times pig-iron of that analysis would be white all the way through? Is there any particular application

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of that chart which would answer why it should be gray at one time and white at another?

MR. STOUGHTON:—I presume Mr. Sweetser has taken into consideration the casting-temperature in each case.

MR. SWEETSER:—I do not know what the temperature is in either case.

MR. STOUGHTON:—I should suppose then that it is a case of casting-temperature or rate of cooling, because the higher the temperature at which it is cast, the more likely it is to be white, and the lower the temperature at which it is cast, the more likely it is to be gray. Where it is just in that intermediate stage where it may be either one, a slight difference in the casting-temperature might push it to either side of the line.

MR. SWEETSER:—Which side of the line have we got to go to get that gray iron? Can you tell me that from the chart?

MR. STOUGHTON:—With 3 per cent. of silicon you ought to get gray iron on the right-hand side of the eutectic iron. The eutectic line, with perhaps 0.3 per cent. of silicon and phosphorus under 0.2, might be under 2.9 per cent.; it might be somewhat lower than that. If you had less than the eutectic amount of total carbon, it would be white, and with more it would be gray.

MR. SWEETSER:—3.9 per cent. of carbon?

MR. STOUGHTON:—I am simply stating that from such investigation as I have made. We have not enough to be certain.

MR. SWEETSER:—I have noticed that in using Pocahontas coke it is possible to make an iron with a very low silicon and still get a gray iron even when cast in chills. And there have been times when the iron would be chilled all the way through to white iron. If there is any way to connect that chart with the question of just what side of the eutectic line the white iron is, it would be a great help to practical furnace-men. And I do want to say here that although that chart is very compli-

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cated to me, I hope some time to learn something about it. A few years ago Dr. Howe gave a tri-axial chart which seemed as complicated to me as this one, but I have since learned that it can be of use in blast-furnace practice; and if this new chart is practical, then we are going a good ways ahead.

MR. JOHNSON:—If Mr. Sweetser will make carbon determinations of the white iron and the gray, I think he will find where the difference comes in. We are more or less familiar with that phenomenon, and frequently get irons almost totally deficient in chill, although the silicon is such that they should have a chill. The thing we have found in regard to them is that they are high-carbon irons, but incidentally they are physically weak and rotten iron.

JAMES F. KEMP, New York, N. Y.:—It may be not entirely without interest to the members of the Institute to know that geologists are working along very similar lines to those set forth in this diagram. This term "eutectic" has become very familiar to us. In analyzing molten rocks we go through processes very similar to those which are employed in the study of alloys. There are many surprising and very close parallels between the structures of the igneous rocks which we find in the field and the structures brought out by the metallographic investigation of alloys.

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Wittorff's Iron-Carbon Equilibrium Diagram.

BY BRADLEY STOUGHTON, NEW YORK, N. Y.

(Cleveland Meeting, October, 1912.)

My attention was called by Prof. Henry M. Howe to the article by N. M. Wittorff, entitled Preliminary Investigation of Primary Crystallization and Subsequent Physico-Chemical Permutations in Iron-Carbon Alloys Containing More Than 4 Per Cent. of Carbon. I thereupon wrote to Mr. Wittorff and received a copy of the paper in the Russian language, with permission from him to publish a translated abstract in our *Transactions*, the original having appeared in the *Journal of the Russian Chemical Society*, vol. xliii. (1911), in which it occupies some 78 pages, with 10 illustrations and 47 micrographs. The novelty of the data presented, and especially the scientific importance of Mr. Wittorff's deductions, as well as their value in helping to clarify our meager understanding of that useful but puzzling commodity, cast-iron, seemed to more than justify a publication of the essential facts in concise form in the English language, which has not been done previously so far as I know, although a very brief abstract has been published in German, and a somewhat more extended one in French.[1] When the work of translation was begun, I did not know that it bore in part upon the investigations of Ruff, which Howe is reviewing at some length at this meeting. This coincidence renders the following condensed translation of Wittorff's work, which I have prepared with the aid of Sidney Marcus as interpreter, the more appropriate here.

The result of Wittorff's study is greatly to extend our knowledge of the hyper-eutectic alloys of iron and carbon, and to add several significant lines at the right-hand side of the diagram originally drawn by Bakhuis-Roozeboom, and developed by Roberts-Austen, Benedicks, Carpenter and Keeling, Upton, Goerens, and others. Some of the original lines of the dia-

[1] *Revue de Métallurgie*, vol. ix., No. 8, pp. 600 to 617 (Aug., 1912.)

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gram, together with the data added by Wittorff, are shown in Fig. 1. So far as he now thinks it advisable to express his views, Wittorff interprets this diagram as follows:

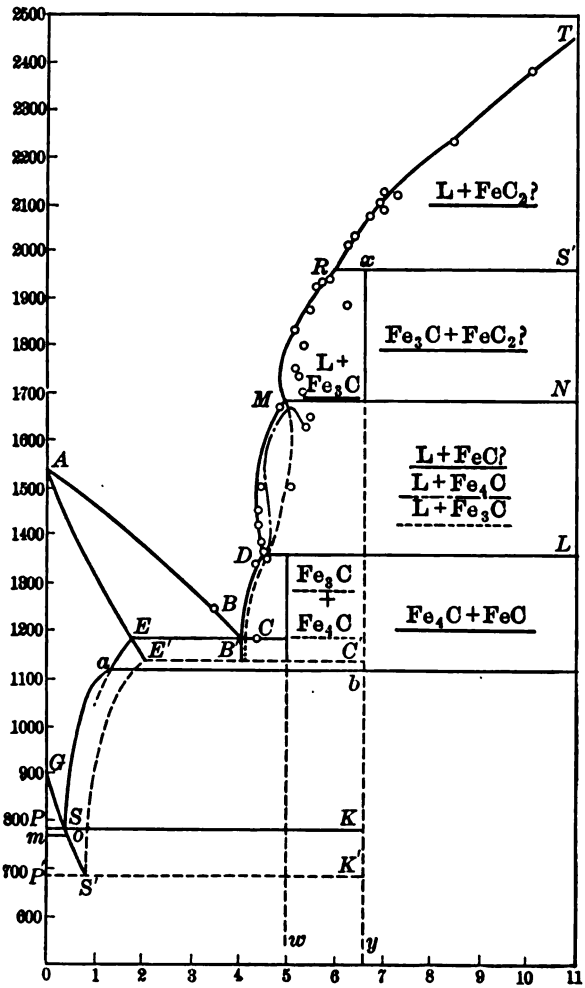

FIG. 1.—WITTORFF'S IRON-CARBON DIAGRAM.

1. Liquid alloys containing carbon in proportion from 10 down to 6.2 per cent. begin to crystallize at temperatures between 2,380° and 2,000° C., by precipitating a highly-carburized phase which seems to be a carbide—FeC_2. This crystallization occurs along the curve *T–R*. Figs. 2 to 5 are micrographs of these alloys, showing the needle-like crystals of this

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constituent. Some of these crystals were studied by Wittorff, and found to be magnetic, to be relatively hard, and to show by analysis 31 and 34 per cent. of carbon, which implies the formula FeC_2 (30 per cent. of carbon). They were of light-gray color and silvery reflection; they etched slowly with a 1 to 4 aqueous solution of nitric acid and were plated with copper when dipped in a very weak solution of copper sulphate. On dissolving them in nitric acid, a dark residue was left.

2. The same constituent (FeC_2?) crystallizes when iron-alloys which are in contact with, and dissolving, graphite, are heated to 2,600° C.

3. At temperatures beginning somewhat below 2,000° and extending down to about 1,700° (see curve *R–M*), the primary precipitations are pure cementite—Fe_3C. (See Fig. 5.) Wittorff's investigations are here confirmed by those of Hanemann. The almost vertical direction of the lower part of the line *R–M* indicates that, in this range of temperature, the molecular composition of the liquid phase changes rapidly, and this change must of necessity be accompanied by a lowering of the concentration of the cementite in the liquid phase of the alloy. Along that part of this curve which inclines in the direction of the carbon ordinate—that is, the part below the point of tangency of a vertical line and the curve *R–M*—there occurs, not a precipitation, but a solution of the previously-formed crystallites of cementite.

4. All the alloys between the lines *R–S'* and *M–N* will contain a small proportion of liquid phase when they average less than 6.67 per cent. of carbon—that is, when they lie to the left of the line *x–y*—and will not contain any liquid phase at all when they contain more carbon than 6.67 per cent. Wittorff explains this condition as follows: Not only does there occur in the liquid phase on the curve *R–M* a stable crystallization of cementite (see paragraph No. 3 above), but the system, $L + FeC_2$ (L is the symbol used throughout in the text to denote the liquid phase), which is normal above the line *R–S'*, undergoes an invariant permutation (r = constant) on the line *R–S'*, as follows:

$$L + FeC_2 \longrightarrow Fe_3C$$

Then the liquid-solid system above the line *R–S'* changes at that line into the conglomerate: $Fe_3C + FeC_2$, or else into the

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FIG. 2. × 100. UNETCHED. 12.65 % C. QUENCHED AT 2,119° C. PRIMARY PRECIPITATION OF FeC_2.

FIG. 3. × 100. UNETCHED. 12.53 % C. QUENCHED AT 2,028° C. LARGE PRIMARY PRECIPITATIONS OF FeC_2 FORMED ON COOLING FROM 2,380° to 2,028° C.

FIG. 4. × 100. ETCHED WITH HNO_3, 1 : 4. QUENCHED AT 2,028° C. 12.53 % C. LARGE PRIMARY PRECIPITATIONS OF FeC_2, ARROWS OF PRIMARY FeC_3 AND GRAPHITE (BELOW TO THE RIGHT).

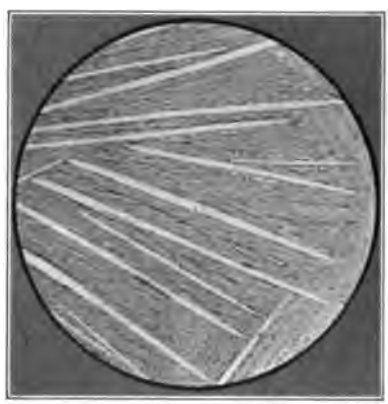

FIG. 5. × 100. ETCHED WITH HNO_3, 1 : 4. ABOUT 6.3 % C. QUENCHED AT 2,028° C. PRIMARY Fe_3C (ARROWS) ON A EUTECTIC-LIKE GROUNDMASS OF Fe_3C + Fe_4C.

FIG. 6. × 100. ETCHED WITH DILUTE HNO_3. CRYSTALLITE FeC_2 SURROUNDED BY Fe_4C PASSING INTO DENDRITES SURROUNDED BY THE EUTECTIC-LIKE

FIG. 7. × 100. ETCHED WITH HNO_3, 1 : 4. QUENCHED AT ABOUT 1,660° C. 6.74 % C. DENDRITES OF Fe_4C IN THE EUTECTIC-LIKE CONSTITUENT

FIG. 8. × 100. ETCHED WITH HNO_3, 1 : 4. .73 % C. QUENCHED AT 1,300° C. ARROWS OF FeC; DENDRITES OF Fe_4C AND EUTECTIC-LIKE CONSTITUENT $Fe_3C + Fe_4C$.

FIG. 9. × 100. ETCHED WITH HNO_3, 1 : 10. 7.4 % C. QUENCHED AT 1,640° C. THE RESIDUE OF THE CARBIDE FeC_2 IS BORDERED BY THE CARBIDE Fe_4C.

FIG. 10. × 100. ETCHED WITH HCl, 1 : 20. 4.4 % C. QUENCHED AT 1,160° C. MUCH Fe_4C; TRACES OF GRAPHITE FORMED FROM DECOMPOSITION OF FeC.

FIG. 11. × 100. ETCHED WITH 4 PER CENT. SOLUTION OF PICRIC ACID. QUENCHED AT 1,120° C. 4.81 % C. ARROWS OF FeC. Fe_4C AND THE BEGINNING OF ITS DECOMPOSITION.

FIG. 12.—SAME AS FIG. 11, BUT DECOMPOSITION OF Fe_4C HAS PROGRESSED FURTHER, BECAUSE QUENCHED AT 1,100° C. 6.73 % C.

complex: L + Fe_3C, depending on whether the average concentration of the carbon is more or less than 6.67 per cent.

5. When the alloys cool through the range of temperatures between 1,650° and 1,330° C.—that is, between the lines *M–N* and *D–L* in Fig. 1—the previously-crystallized mass begins to liquefy. This liquefaction is particularly noticeable in alloys containing about 7 per cent. of carbon (observations of Wittorff and Moissan). (It is to be noted that Fe_3C contains 6.67 per cent. of carbon.) The liquefaction commences at the line *M–N* and is a maximum at 1,370°. Wittorff studied the liquefaction of alloys of widely-differing carbon-concentrations, and obtained the results given in Table I.

TABLE I.—*Temperature of Liquefaction of Alloys.*

Amount of Carbon.	Temperature of Noticeable Liquefaction.	Temperature of Maximum Liquefaction.
About 7 per cent.	About 1,630° C.	1,340° C.
More than 5 per cent. / Less than 6 per cent.	.....	1,336°
6.0 per cent.	1,550°	.....
(Fe_3C) 6.67 per cent.	1,600°	.....
7.73 per cent.	1,600°	1,370°
About 17.0 per cent.	Less than 1,700° / More than 1,600°	.....
6.73 per cent.	1,640°	.....

Wittorff ascribes the liquefaction to the decomposition of Fe_3C into a liquid and a solid phase. He collected crystals of this solid phase and learned by analysis that it contained 17.57 per cent. of carbon, whence he states, with reservations, that it consists of the iron carbide, FeC (which contains 17.65 per cent. of carbon). The reaction represented by the line *M–N* would then be:

$$Fe_3C \longrightarrow L + FeC.$$

The liquefaction is then due to the exothermic decomposition of Fe_3C, or, in other words, to its endothermic formation.

Upon the decomposition of FeC into L + FeC, the quantity of the liquid phase, L, constitutes about 86 per cent. of the mass of the complex, L + FeC. Therefore the liquefaction will be particularly noticeable in alloys containing more than 6.67 per cent. of carbon, because such alloys will be entirely solid down to the temperature of the line *M–N*, but below that

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line the quantity of their liquid phase will exceed by 5 or 6 times the quantity of their solid constituent, FeC. Indeed, liquefaction gives the appearance of being almost complete unless the carbon exceeds 8 per cent.

6. When we heat alloys into the area *L–D–M–N*, and then stir the melted mass, there swim to the surface thin plate-like crystals which Wittorff has investigated and believes, with reservations, to be iron carbide, FeC. Such alloys, rapidly cooled from about 1,600° C., show under the microscope these crystals of FeC (?) as long arrows with needle-like shoots. (See Figs. 5 to 8. The crystals are etched weakly with acid; they are darker than the crystals obtained above 2,000° (FeC_2?), but lose their color when rubbed with chamois wet with alcohol.)

7. After the complete disappearance of the primary cementite, then the crystallization of FeC continues along the curve *M–D* down to the temperature of the outermost point of this curve—that is, the point at which a vertical tangent would touch this curve. Below that point, FeC crystals begin to dissolve, and at the temperature of the line *D–L*, a reaction occurs between the liquid phase and the FeC, which deposits a new carbide, Fe_4C, according to the equation following:

$$L + FeC = Fe_4C.$$

This operation completes the crystallization (freezing) of alloys containing more than 5 per cent. of carbon, and they thus pass into a two-phase solid conglomerate, the basic mass of which is formed by Fe_4C and is pierced by arrows of FeC, the number of which increases with the average concentration of carbon. (See Fig. 9.)

If the average carbon-content is not greater than 5 per cent., the complex: L + FeC, normal above the line *D–L*, changes at that line into L + Fe_4C.

8. Alloys quenched below 1,700° always contain a new constituent. This constituent surrounds arrows of FeC (?) or of FeC_2 (?), and forms dendrites on a eutectic-like background. (See Figs. 6, 7, 8, 9.) This constituent Wittorff holds to be Fe_4C. It is distinguished from cementite by being etched with very dilute nitric acid (1 : 4). It is not noticeably etched by a 4 per cent. alcoholic solution of picric acid, nor by an aqueous solution of sulphuric acid (1 : 10 by weight), unless the alloy is quenched below 1,160°. With an increase of concentration of

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carbon from 4.1 to 5.0 per cent., the quantity of primary precipitation of this constituent (Fe_4C) grows from 0 to 100 per cent. (Fe_4C corresponds to 5.02 per cent. of carbon.) When an alloy contains from 6 to 7 per cent. of carbon, the basic mass of the metal consists of this constituent, and the remainder of arrows of FeC (?).

9. Solidification ends at the temperature corresponding to the level of the line *E–B–C* (1,180° C.). At this point the stable eutectic of the series, containing 4.1 per cent. of carbon, crystallizes into gamma austenite and the carbide, Fe_4C. (See Fig. 10.)

10. At the temperature of the line *a–b* (*i.e.*, below 1,130°, and probably approaching 1,100°), Fe_4C decomposes into gamma austenite and graphite. Wittorff bases this assertion on his own investigations, and on those of Goerens and Gutowsky. (See also Figs. 11, 12.)

11. Primary graphite originates through the dissociation of the phase crystallizing above 2,000° (FeC_2?), and the phase crystallizing in the interval of temperature between 1,600° and 1,400° (FeC?). The conditions of dissociation of these carbides have not yet been cleared up, and therefore no lines have been drawn for them on the diagram. It can only be said that the dissociation of FeC_2 must be no higher than the line *R–S* and no lower than the line *M–N;* and that of FeC must be no higher than the temperature of the line *D–L*. Otherwise, graphite would have separated from the liquid alloy, which has never been observed.

Observations on the Conduct of the Experiments.

Wittorff cites the following statement of Moissan:[2] "When we heat pure iron in contact with pure carbon in an electric furnace, its fluidity becomes less and less as the temperature becomes higher. At the melting-point of titanium" (which Moissan has elsewhere stated to be above 2,500° C.) "the content of the crucible has become so viscous that it will not flow easily. If then the temperature is lowered, the bath regains its fluidity and flows like water." Wittorff formed the conclusion that the cause of the solidification of the iron-alloys

[2] *Comptes rendus de l'Académie des Sciences*, vol. cxxiv., No. 14, p. 716 (1897).

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at very high temperatures was the crystallization of an endothermic iron carbide, and he resolved upon a systematic study of the hyper-eutectic alloys of iron and carbon. He found a kryptol furnace satisfactory up to temperatures of 2,000°; above that heat he found an arc-furnace such as shown in Fig. 13 to be the best adapted. A Féry pyrometer was used, and found to be accurate within a few degrees at the highest temperatures.

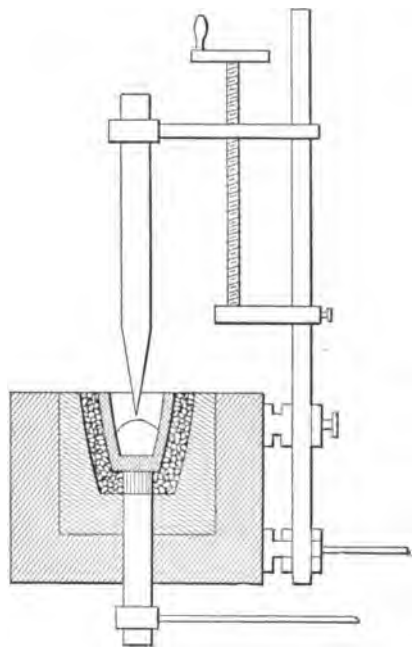

FIG. 13.—WITTORFF'S ELECTRIC-ARC FURNACE.

His first observations confirmed those of Moissan, namely, that iron heated in contact with carbon and maintained slightly above 2,000° C. rapidly dissolves the carbon and soon becomes entirely solid. On quenching such a solidified melt from 2,119° in snow-water, he found that it contained 12.65 per cent. of carbon. (See Fig. 2.) A similar mass which had been allowed to solidify at the high temperature became liquid upon cooling to 1,370° C. As has already been stated, Wittorff's investigations convinced him that the cause of the solidification at high temperatures was the formation of the endothermic carbide,

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Fe_3C, and the liquefaction at lower temperatures was due to the dissociation of this cementite into a liquid phase and FeC.

Wittorff's monograph also includes a discussion of the metastable changes of the alloys, and the work of Goerens, Goerens with Gutowsky, and Hanemann, but the inclusion of these remarks would take us beyond the scope of this brief abstract of a very valuable paper. His paper also comprises a discussion of those results obtained from Ruff which seem to contradict his own deductions:

Observations on Ruff's Equilibrium Diagram.

Wittorff's work had already been finished when he learned of the paper published by Ruff.[3] Ruff investigated the solubility of carbon in iron at temperatures from 1,200° up to 2,630° C. In two major respects Ruff disagrees with the independently-formed conclusions of Wittorff:

Ruff concludes that the metal between 1,135° and 2,600°, saturated with carbon, precipitates graphite during a stable process of cooling, instead of carbides, Fe_3C, FeC, and Fe_2C, the existence of which he admits only in the liquid alloys. In the second place, Ruff found that the rising curve, corresponding to the line *R–T* in Fig. 1, turns sharply to the left at a point represented by the temperature of 2,200° and carbon-concentration of about 9.6 per cent., and, reversing its direction, comes to a point represented by a temperature of 2,626° and carbon-content of 7.7 per cent.

As to the first of these assertions—that graphite is precipitated on cooling, instead of carbides—Ruff bases the deduction chiefly on the observation that the separated crystals swim to the surface, and that they may be pressed into a compact mass which swims to the surface. These arguments are not conclusive, in Wittorff's opinion, because he has observed that even the comparatively dense cementite crystallites swim to the surface when the upper layers of the liquid are stirred, and he found that graphite crystals are not cohesive and cannot be pressed into a compact mass.

As to the turning of the equilibrium-curve to the left above 2,200°, Wittorff explains Ruff's results on the ground that

[3] *Metallurgie*, vol. viii., No. 14, p. 417; No. 15, p. 456; No. 16, p. 497 (July 22, Aug. 8, Aug. 22, 1911).

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Ruff's quenching of his specimens was not sufficiently rapid to catch the unusually swift crystallization of FeC_2, and that therefore what Ruff determined in his specimens quenched above 2,200° was not the average composition of the whole alloy, but the composition of the matrix from which the FeC_2 had separated.

Ruff had explained the turning to the left of the curve with rising temperature upon the basis of the following hypothetical decomposition:

$$Fe_2C \rightarrow 2\ Fe + C.$$

This is impossible, according to Wittorff, because it calls for an exothermic synthesis of Fe_2C, which contradicts Ruff's own equation of dissociation, as follows:

$$2\ Fe_2C \rightarrow 2\ Fe_3C + C \text{ and } Fe_3C \rightarrow 3\ Fe + C.$$

Now, if we multiply the second equation by 2 and then add them together, we get:

$$3\ Fe_2C \rightarrow 6\ Fe + 3\ C.$$

This proves that the synthesis of Fe_2C is endothermic, because the last three reactions correspond to a lowering of temperature.

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The Microstructure of Iron and Steel.

BY WILLIAM CAMPBELL, PH.D., SC.D.,* NEW YORK, N. Y.

(Cleveland Meeting, October, 1912.)

THE structure of iron and steel, though the object of so much study and research for the past 25 years, is by no means thoroughly understood.

In the first place, we have three or more distinct iron-carbon thermal diagrams, each with its own supporters, and each representing a theory of the structure and constitution of cast-iron.

In the second place, by reason of the fact that austenite, the solid solution of carbon in iron, decomposes in the solid state into ferrite, or pure iron, and cementite (Fe_3C), and that quenching from above this transition-range, no matter how rapid, cannot entirely prevent the change and thus preserve the austenite by itself at ordinary temperatures, it happens that under the microscope we find, in addition to the ferrite, Fe_3C, and austenite, which the thermal diagram of steel requires, a series of transition-products with distinctive structures, to which have been given the names martensite, troostite, sorbite, etc. Now these transition-products, in addition to the solid solution, austenite, have been the source of much controversy, at times very bitter. One side would insist that, say, troostite was an entity, which the other side would deny, pointing out that it must be a transition-mixture, with a particular structure. That austenite was a solid solution of carbon in iron has been denied by one school; we have been told that $Fe_{24}C$ dissolved both Fe_3C and Fe, and thus gave rise to a whole series of quenching-products—all this in spite of the fact that no matter how the thermal diagrams differ in regard to cast-iron, they all agree in regard to the solid solution, austenite, for $Fe_{24}C$ finds no place on the diagram.

Thus it happens that while the authorities fight both in regard to the iron-carbon diagram and the structures met with in hard-

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ened, tempered, and certain alloy-steels, the engineer who would apply metallography to every-day problems is bewildered at the outset by the contradiction of axioms, postulates, and fundamental propositions; and unless he realizes that for him these things are relatively unimportant, he will probably give up metallography in despair. After all, if, under the microscope, we can identify certain structures with certain properties; austenite with strength and toughness and want of magnetism; martensite with extreme hardness, brittleness, and want of ductility; sorbite with strength and ductility, though not hardness; pearlite, the mechanical mixture of Fe and Fe_3C, with ductility, but less strength and hardness than the rest, we have a valuable aid in testing material, even if we cannot define with exactness the ultimate constitution of the different things we recognize under the microscope.

In the present paper an attempt has been made to set forth a few of the main features encountered in the microscopic examination of iron and steel, together with some of the points still under discussion.

Classification of Iron and Steel.

I. Wrought-iron:
   1. Puddled iron.
   2. Charcoal-hearth iron (wrought-steel).
   3. Fagoted iron.
   4. Busheled iron.

II. Low-carbon steel:
   1. Pipe-steel, comparison with iron. Welds.
   2. Mild-steel, 0.1 per cent. of C. Segregated bands. MnS, etc.
   3. Structural steel, from 0.2 to 0.3 per cent. of C. Size of grain. Lamination.

III. Medium-carbon steel:
   1. Castings. The diagram. Heat-treatment to refine. MnS.
   2. Axles. "Recrystallization." Heat-treatment.

IV. Rail-steel:
   1. Amount of pearlite. Grain-size. Old rails and new.
   2. Segregation and carbon-content. Flaws.
   3. Tire-steel.

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V. High-carbon steel:
   1. 0.75 to 2.0 per cent. of carbon. Normal structure. Effect of reheating.
   2. Graphite precipitation.
   3. Hardening structures. Tempering.

VI. Cement-steel and case-hardening steel.

VII. Alloy-steel:
   1. Pearlitic: Nickel-steel.
   2. Austenitic: Manganese-steel.
   3. Martensitic: Air-hardening steel.

I. *Wrought-Iron.*

Wrought-iron is the product of the treatment of pig-iron in the puddling-furnace or charcoal-hearth, of reheating bundles of scrap-iron and steel (fagoting), of treating miscellaneous scrap-steel in the puddling-furnace (busheling).

There can be distinguished four main types of wrought-iron: 1, puddled iron; 2, charcoal-hearth iron (wrought-steel); 3, fagoted iron; 4, busheled iron.

1. *Puddled Iron* contains (unlike steel, etc.) no manganese, high phosphorus (which is mainly in the slag), as compared with charcoal-hearth iron or ingot-iron; and under the microscope is composed of polygonal grains of ferrite, through which are distributed threads or elongated flakes of slag, Fig. 1. When one considers the process of manufacture, the cause of this structure becomes evident. The metal comes to nature as solid grains and dendrites of pure iron, which are worked up into a ball, and withdrawn from the furnace dripping with molten slag. Next the greater part of the slag is squeezed out under the hammer, or in the squeezer, while a further slight amount is extruded in the puddle-rolls. The product is a muck-bar of coarse grain and coarse slag-fibers. This is cut up, piled, reheated to welding-heat, and rolled out once more. The resulting metal has proportionately finer grains and finer slag-fibers. A second piling, reheating, and rolling will produce a still finer slag-fiber. In addition, a certain amount of oxide also is introduced as fibers and flakes with each piling and reheating.

An examination of the slag of wrought-iron shows that in the

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case of rounds and squares the form is that of fibers, while with flats and sheets it is necessarily that of flakes and elongated plates. The slag is composite and consists of a dark ground-mass (probably silicate) full of dendrites and round globules of a lighter material, showing a bluish tinge by vertical light (probably oxide). The ferrite matrix is composed of polygonal grains, Fig. 13, the size of which depends on the time and temperature in the reheating-furnace and the amount of reduction of the metal with rolls, presuming that the finishing-temperature is above the critical-point, Ar_3, or, say, 900° C. If, on the other hand, the metal is finished below this point, the grains of ferrite are more or less elongated in consequence. As three different types, we might compare the structure of a piece of large shafting or an old wrought-iron rail with that of a piece of pipe-skelp, and a piece of cold-rolled iron.

2. *Charcoal-Hearth Iron* (*Wrought-Steel*) contains no manganese (being thus distinguished from steel); the phosphorus-content is much lower than that of puddled iron, and the presence of slag distinguishes it from ingot-iron.

In structure, charcoal-iron is very similar to puddled iron, but, owing to the fact that it comes in contact with the glowing bed of charcoal during manufacture, parts of the bloom become carburized, and these appear in the finished sheet or bar as bands of ferrite, containing more or less pearlite. The center of the band contains most pearlite, which gradually fades away towards the outside and passes into pure ferrite. This is typically shown in Fig. 2. The slag has the same characteristics as in puddled iron.

Dannemora iron is converted into blister-steel by cementation, and may be used by remelting in crucibles for the manufacture of crucible-steel. Hence the presence of considerable quantities of carbon is not objectionable. On cutting a cross-section through certain brands, the banded appearance shows very markedly to the eye. Under the microscope we find alternating bands of ferrite and slag, ferrite and pearlite, and even pearlite alone, the one band passing imperceptibly into the next. Such material has been called wrought-steel, when capable of hardening.

3. *Fagoted Iron,* as commonly made, is a mixture of old

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wrought-iron and scrap-steel, preferably low in carbon. When the scrap is small, a "box-pile" is made, with base and sides composed of sheet wrought-iron, and the interior is filled with a mixture of iron and scrap-steel. The whole is heated to welding-heat and then rolled out, as in the manufacture of puddled iron. Hence this material usually contains an appreciable amount of manganese. In structure it resembles charcoal-hearth iron, for it contains bands of ferrite and pearlite; but these bands, due to the scrap-steel, do not pass imperceptibly into the ferrite (and slag) areas as in charcoal-hearth iron. There is generally a sharp line of division between the two, with a more or less continuous band of slag and oxide fibers, which is seldom the case in charcoal-hearth iron. Fig. 14 is a cross-section of such material.

4. *Busheled Iron.*—By busheling scrap is meant the charging of the puddling-furnace with baskets of small steel scrap, bringing them to incipient fusion, balling the mass, and squeezing out and rolling as in the true puddling process. This produces a bar or sheet composed of particles of steel compactly welded together, with intermingled slag, closely resembling puddled iron, but containing much more manganese (and generally less phosphorus). The structure of this material is extremely hard to distinguish from puddled iron on the one hand and charcoal-hearth iron on the other, because the distinct lines of separation between iron and steel as formed in fagoted material are missing.

On the other hand, if the steel scrap is actually melted in the puddling-furnace, there is nothing in etching to distinguish the structure of the finished product from wrought-iron.

II. *Low-Carbon Steel.*

For the sake of convenience there is included in this group material up to 0.3 per cent. of carbon, such as is used for structural purposes.

1. *Pipe-Steel and Skelp.*—The chief question in microscopic examination is to decide whether a pipe is composed of puddled iron, charcoal-hearth iron, or steel. The following are analyses of these three materials, together with ingot-iron for comparison:

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	Mn.	P.	S.	Slag.	Si	C.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Puddled iron.....	Trace — 0.2	0.12	0.02	1.0	0.2	0.03
Charcoal-iron.....	Trace.	0.03	0.007	0.75	0.04	0 — 0.09
Steel	0.35 — 0.45	0.1	0.04	0.1	0.04	0.07 — 0.09
Ingot-iron.....	0.02	0.005	0.015	.....	0.005	0.02

The carbon averages 0.03 per cent. in all but steel. Of course, there are many variations from the above, and many cases cannot be decided by analysis or by the microscope alone. We have already discussed the difference between puddled and charcoal-hearth iron. Steel is composed of ferrite with from 8 to 10 per cent. of pearlite, usually occurring as small patches between the grains of the ferrite. In addition, there are small elongated globules of dove-colored manganese sulphide and occasionally thinner black threads of slag or oxide. The structure of both the ferrite and the pearlite differs in skelp, sheet, small and large pipe. In the sheet the ferrite is usually in polygonal grains of small size, in small pipe the grains are somewhat larger, but still more or less polygonal, while in large pipe this feature is missing as a rule, and the ferrite grains are more or less mixed up with the pearlite scattered in and between them. This is owing to the temperature and the time given in the heating-furnace for welding. Again, in sheet and small pipe the pearlite often shows marked segregation, the ferrite and the pearlite having coalesced with the main mass of the ferrite, while the cementite forms small irregular patches. The same is very common in low-carbon steel rounds of about 0.1 per cent. of C. Light etching with picric acid shows up the pearlite, while prolonged etching will show up the grain of the ferrite also. Picric acid seems preferable to nitric or sulphuric acid, because it yields sharper boundaries to the ferrite grains and does not eat into the material around slag, MnS, or pearlite.

Macroscopic etching is commonly used to distinguish between steel and wrought-iron pipe. After the section has been filed smooth and rubbed down with Nos. 0 and 00 emery-paper, it is deeply etched with nitric, sulphuric, or other etching-medium. The steel shows up plain, while the wrought-iron shows a concentric structure, due to the inclosed slag.

Lap-welds and butt-welds can usually be distinguished in the

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etched specimen by the eye. Under the microscope the lap-weld is seen to form an obtuse angle with the lamination of the steel or wrought-iron and can be traced by entrained particles of oxide or cinder. The butt-weld can be traced much more readily; for at the outside and the inside of the pipe there is usually a slight V-shaped depression, the bottom filled with oxide, which indicates the direction of the weld. In the metal between these two points a few globules of oxide indicate the weld, which can be recognized also (1) in steel because the pearlite laminations are not continuous; (2) in wrought-iron because the slag globules and threads are discontinuous and often bent over by the force of the weld.

The structure of ingot-iron consists of polygonal grains of ferrite, as in Fig. 13.

Mild-Steel.—Steel with about 0.1 per cent. of C is widely used for light shafting, rods, bands, rivets, spikes, screws, and machine-work.

The analyses vary, depending on whether it is Bessemer or basic open-hearth.

Under the microscope we find it composed of ferrite, with about 10 per cent. of pearlite. The grain-size depends chiefly on the cross-section, or, rather, on the amount of reduction in rolling from the last heating. As in the case of pipe-steel, so here we often find the pearlite segregated into little globules, or flakes of cementite. In a longitudinal section the pearlite generally shows some lamination, which is also seen by the way the MnS (and slag) is elongated. Fig. 3 is a longitudinal section of a 2.5-in. round. The small black patches are pearlite, the long, thin streak across the center is slag. Fig. 4 is a section of a 3.5-in round, and the dark etching band across the center is a "ghost," and owes its presence to segregation of P, MnS, etc., which produces "streaky" steel. In screw-stock the MnS is very marked in the longitudinal sections.

3. *Structural Steel.*—As the carbon is increased from 0.2 to 0.3 per cent. there is a proportionate increase in the pearlite, which is more commonly laminated with the ferrite. Streaks of MnS and occasionally of slag are seen, usually imbedded in bands of ferrite. As before, the slag is black and in much thinner threads than the MnS, because its freezing-point is lower and the plastic range longer, and therefore it tends to become

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more strung out in the rolling. The rule generally holds good that the thicker the section the coarser the grain, and the less pronounced the lamination. Fig. 7 is a section of a 2- by 0.25-in. piece, showing the lamination.

III. *Medium-Carbon Steel.*

1. *Castings.*—With regard to the structure of steel with from 0.3 to 0.4 per cent. of carbon which has been rolled, the above remarks hold good. The structure of castings, however, is quite different. When a casting freezes, dendrites of the solid solution austenite tend to grow perpendicular to the cooling-surface, and we get the characteristic "pine-tree" crystallization on the outside. The interior is made up of more or less granular crystals. On cooling down there is no change until the temperature reaches the upper critical point, $Ar_{2\text{-}3}$, or the line *GOS* of the iron-carbon diagram,[1] Fig. 25, when ferrite commences to separate out around and in the grain of the austenite, giving the characteristic austenitic structure resembling the Widmanstätten figures of meteorites, or, if the carbon be high, the feathery ferrite. On reaching the lower critical point, Ar_1, or *PSK* of the diagram, from 700° to 660° C. (depending on the manganese), the remaining austenite, which now contains approximately 0.9 per cent. of carbon, splits up into the eutectoid pearlite, the mechanical mixture of ferrite and cementite. The structure of this pearlite and perhaps its content of carbon depend largely on the rate at which the steel passes through this critical temperature. Rapid cooling in air, for instance, yields very fine pearlite, while very slow cooling gives us the coarsely laminated variety whose pearly appearance is marked to the naked eye. Fig. 9 shows a steel with C, 0.35, and Mn, 0.66 per cent., as cast.

Such coarsely crystalline material is brittle, and must be annealed, or, more properly speaking, refined. This is done by reheating to above the upper critical point, $Ac_{2\text{-}3}$; that is, just above *GOS*. On passing the line *PSK*, or Ac_1, the pearlite changes over into austenite, and with increase in temperature the solid solution austenite dissolves more and more of the ferrite until, on the $Ac_{2\text{-}3}$ change being completed, we have

[1] See Heyn. Report on Progress of Metallography, *Proceedings of the International Association for Testing Materials*, vol. ii., No. 11 (1912).

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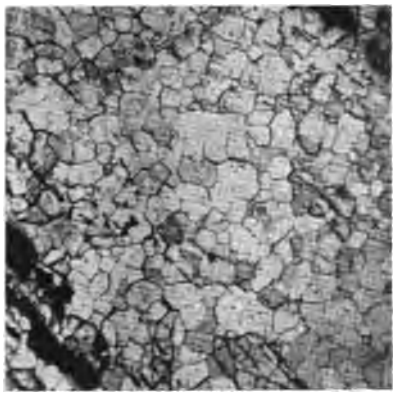

FIG. 1.—WROUGHT-IRON.
MAGNIFIED 50 DIAMETERS.

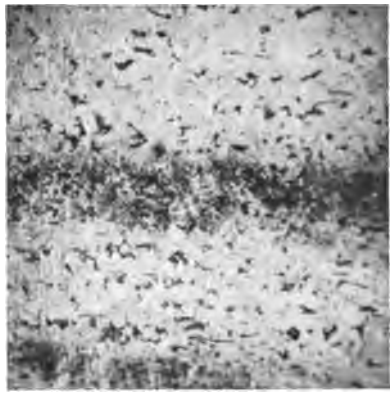

FIG. 4.—LOW-CARBON STEEL.
"GHOST." MAGNIFIED 50 DIAMETERS.

FIG. 2.—CHARCOAL-HEARTH IRON.
MAGNIFIED 50 DIAMETERS.

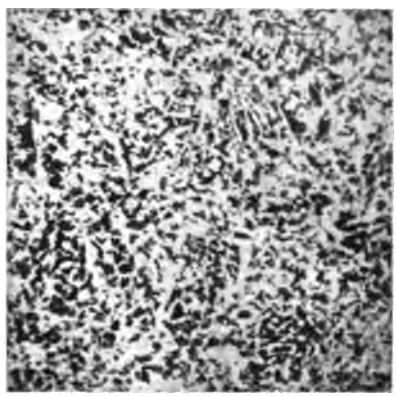

FIG. 5.—REFINED STEEL CASTING.
MAGNIFIED 50 DIAMETERS.

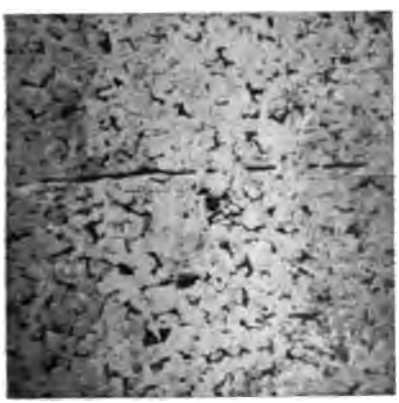

FIG. 3.—LOW-CARBON STEEL.
MAGNIFIED 50 DIAMETERS.

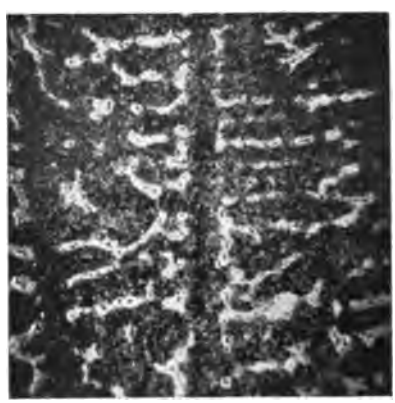

FIG. 6.—MnS, ETC., IN REFINED
CASTING. MAGNIFIED 30 DIAMETERS.

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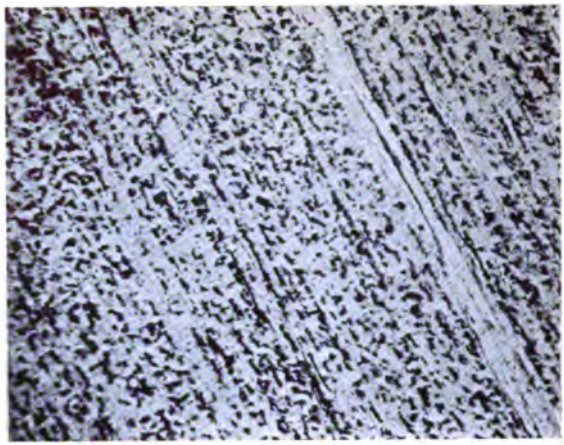

FIG. 7.—STRUCTURAL STEEL. MAGNIFIED 50 DIAMETERS.

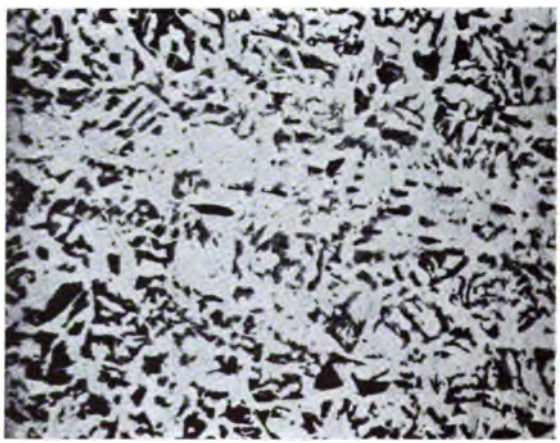

FIG. 8.—3.5-IN. ROUND. MAGNIFIED 50 DIAMETERS.

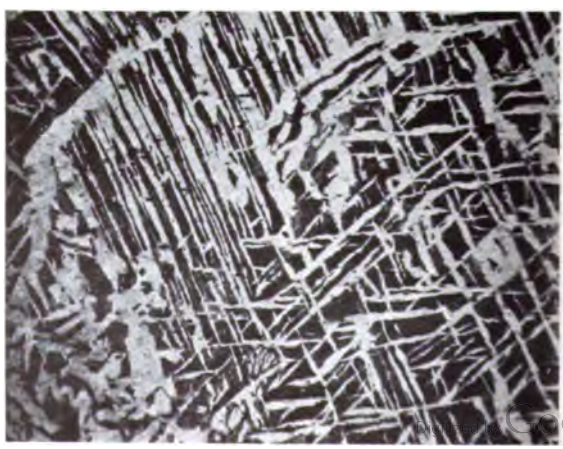

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FIG. 9.—STEEL AS CAST. MAGNIFIED 50 DIAMETERS.

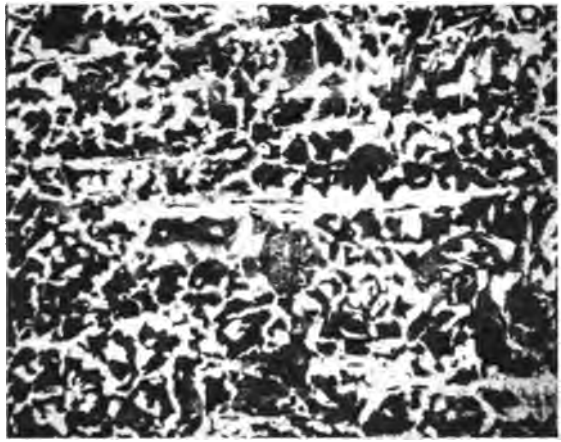

FIG. 10.—3.5-IN. SHAFT. MAGNIFIED 50 DIAMETERS.

FIG. 11.—RAIL-HEAD. ETCHED.

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FIG. 12.—BROKEN SHAFT. "FATIGUE."

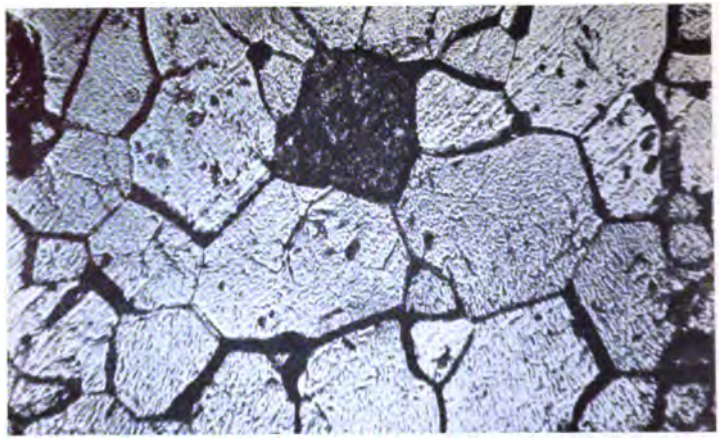

FIG. 13.—WROUGHT-IRON. MAGNIFIED 500 DIAMETERS.

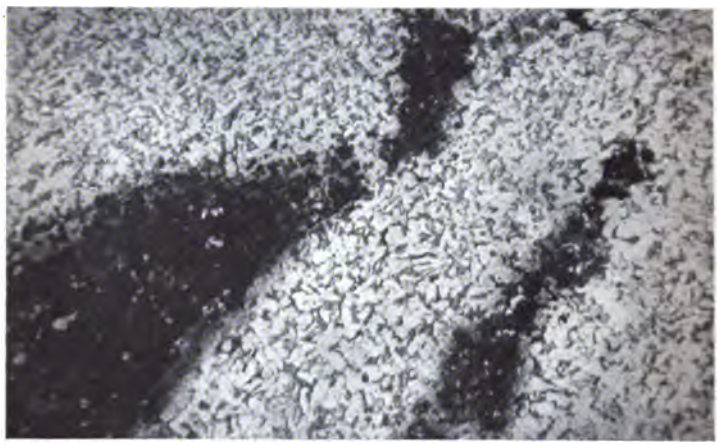

FIG. 14.—WROUGHT-IRON MADE FROM SCRAP. MAGNIFIED 70 DIAMETERS.

FIG. 15.—2 PER CENT. C. HEATED TO 1,070° C. MAGNIFIED 40 DIAMETERS.

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FIG. 16.—2 PER CENT. C. HEATED TO 1,200° C. MAGNIFIED 40 DIAMETERS.

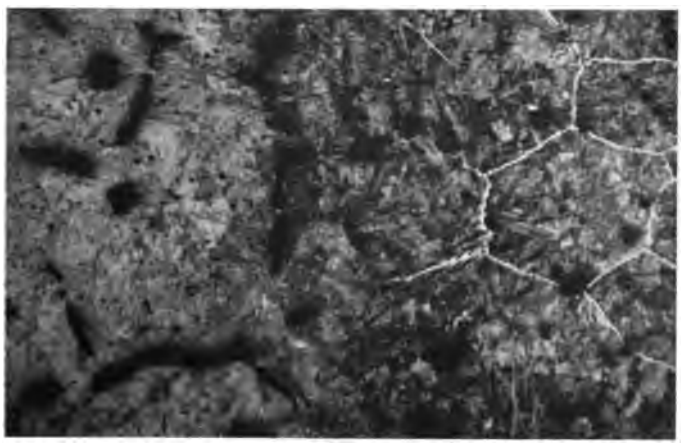

FIG. 17.—GRAPHITE AND CEMENTITE. MAGNIFIED 60 DIAMETERS.

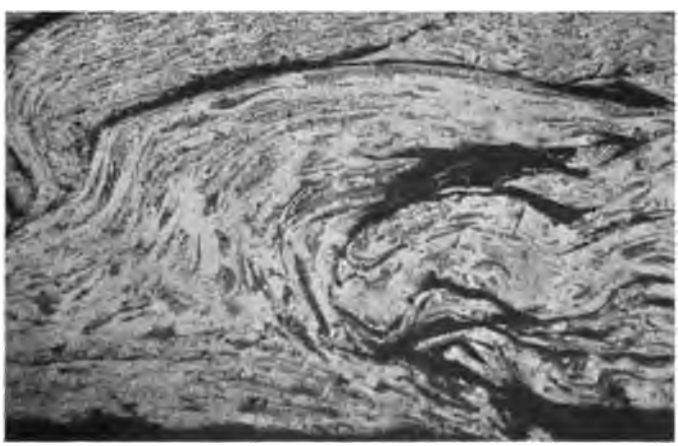

FIG. 18.—TIRE, SHELLING OUT. MAGNIFIED 70 DIAMETERS.

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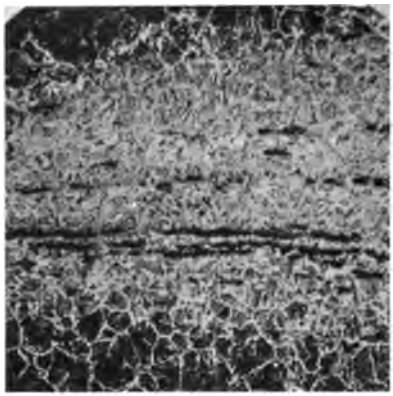

FIG. 19.—SLAG IN RAIL-STEEL. MAGNIFIED 50 DIAMETERS.

FIG. 22.—AUSTENITE AND TROOSTITE. MAGNIFIED 50 DIAMETERS.

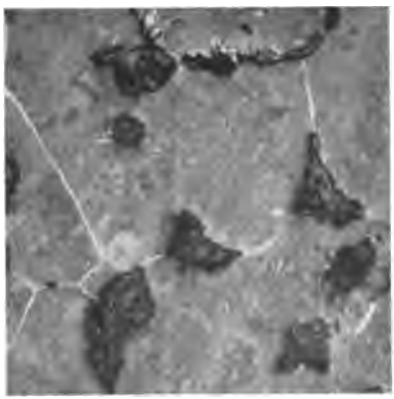

FIG. 20.—TIRE-STEEL, BURNED. MAGNIFIED 60 DIAMETERS.

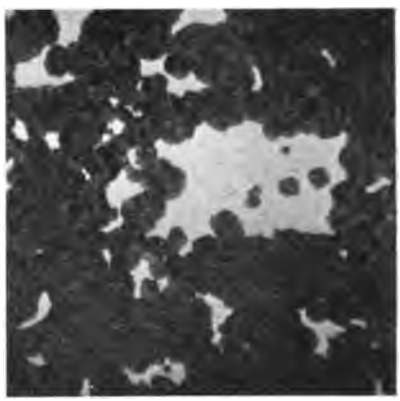

FIG. 23.—TROQSTITE AND AUSTENITE. MAGNIFIED 50 DIAMETERS.

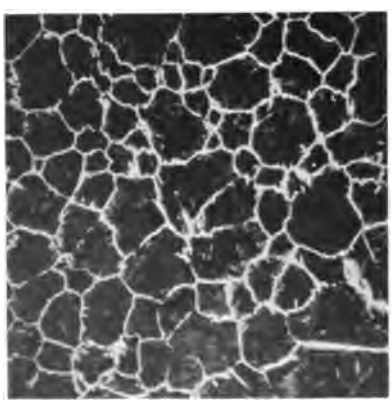

FIG. 21.—CEMENTITE AND PEARLITE. MAGNIFIED 60 DIAMETERS.

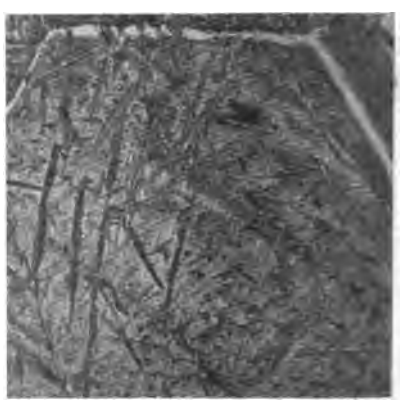

FIG. 24.—CEMENTITE AND MARTENSITE. MAGNIFIED 250 DIAMETERS.

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nothing but fine-grained austenite. On cooling down again, the ferrite separates out as before; but now it is confined to very small grains of austenite, and hence the resulting structure is fine. Fig. 5 shows the same steel heated to 850° C. and thus refined.

It is not uncommon to find that such a treatment has not completely refined a specimen. The reason lies in the presence of globules and films of MnS or oxides, which often assume

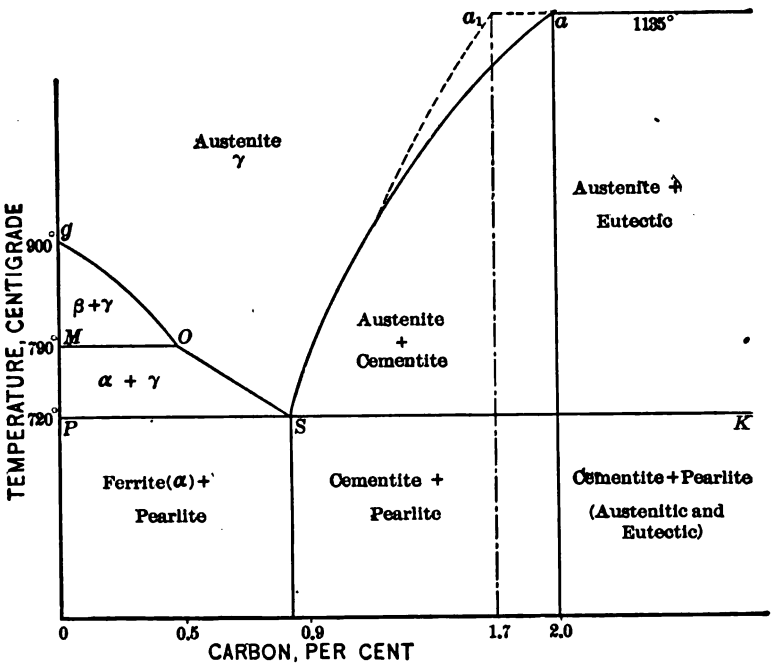

FIG. 25.—AUSTENITE DIAGRAM.

a dendritic form, due to their being deposited in the dendrites of the original casting. Heating to above $Ac_{2\text{-}3}$ truly refines this steel, but on cooling down again the ferrite tends to precipitate on these films and globules of MnS, and lines of weakness result. However, the heat-treatment has materially improved such material, as the tensile tests will show. Fig. 6 is a piece of steel with C, 0.5, and Mn, 0.68 per cent., after refining, and shows the dendritic structure due to manganese sulphide, etc.

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2. *Axle-Steel.*—The structure of axle-steel is similar to that of structural steel, allowing for the increase in pearlite with increase in carbon. Grain-size increases with section. A great many cases of failure have been ascribed to fatigue or recrystallization due to vibration. We see a fracture the outer part of which is very fine and the inner core comparatively coarsely crystalline, as in Fig. 12. On cutting a section from center to outside, the grain-size is found to be practically the same. Of course, there has been no growth of grain. Under repeated strain, a tiny crack starts at the outside and slowly works its way round the section, becoming deeper and deeper. As it progresses, the two faces pound on each other and tend to make the surface smooth. In time the cross-section of the solid metal becomes so small that it can no longer bear the strain, and rupture takes place, giving a normal crystalline fracture. Fig. 8 is a longitudinal section of a 3.5-in. shafting which broke with so-called recrystallization. The structure is the same from outside to center. Through the center of the photograph is a band comparatively rich in ferrite, with characteristic elongated globules of MnS running through it.

Fig. 10 shows the structure of a 3.5-in. shaft which also broke. The carbon-content is much higher, but here again is MnS set in a strip of ferrite.

Thus there are two kinds of fracture: the inside, or normal, bearing a definite relation to the grain-size, and the outside, or abnormal, having no relation whatever to the internal structure of the piece. It is to be noted, however, that such broken material often shows a coarse grain, or much segregation with MnS and associated high-phosphide areas, or both; and it has been proved that reheating to the refining-point and cooling in air, or, better still, quenching and again reheating to from 500° to 600° C., gives material which withstands "fatigue" extremely well.

Axle-steel may therefore be divided into three main groups: forged, "annealed," and "tempered."

The forged material stands up well in use if in perfect alignment, that is to say, if it receives a minimum amount of vibration, or if it is not subject to "fatigue." If the axle or shaft is not properly set, fatigue sets in. It is found that the coarser the grain the more liable are slip-lines to occur, and, therefore,

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fatigue comes sooner, with a tendency to the progressive fracture we have already described.

With the "annealed," or, more correctly speaking, refined, a fine grain is less likely to produce slip-lines, and the material is therefore less liable to fatigue, other things being equal.

The last group, which has been classed as "fibered" material, especially in nickel-steel, withstands strain and vibration best. If we heat to the refining-temperature, $Ac_{2\text{-}3}$, and quench, we get material made up of extremely fine austenite, etc. On reheating to from 500° to 600° C. there is a great change within this tiny grain itself. If we fracture a longitudinal section it has a distinctly fibrous appearance. It is much finer than the structure found in wrought-iron. This fibrous appearance is due entirely to the method in which the individual grains elongate and break; it is accentuated by traces of MnS, etc., which have been elongated in the direction of rolling.

This material has about the same size of grain as that which has been refined, but the interior of the grain is quite different in the latter. The pearlite is comparatively coarse and, under high power, is distinctly pearly, that is to say, the ferrite and cementite can be resolved. The tempered material is pearlite, but in a much finer condition. It is almost sorbitic; for it is the product of the change from austenite through martensite with troosite into sorbite and pearlite, which takes place on reheating. Such material has been called "anti-fatigue;" and nickel-steel, giving the maximum amount of fibrous structure, is ideal for such material.

IV. *Rail-Steel.*

With an increase in carbon to from 0.4 to 0.6 per cent. there is a further increase in pearlite. In estimating the amount of carbon by the microscrope, both the amount of manganese (up to 1 per cent.) and the comparatively rapid cooling in air have to be taken into account. Thus, a low-carbon steel with about 10 per cent. of pearlite will contain about 0.09 per cent. of C, the factor being 0.9. With from 25 to 30 per cent. of pearlite the factor is about 0.8 and there is, say, from 0.20 to 0.25 per cent. of C. At 50 per cent. of pearlite, multiplying by 0.75 gives about 0.37 per cent. of C. With 75 per cent. of pearlite and a factor of 0.7 there is about 0.42 per cent. of C; while at 0.6 of C there is

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only a trace of ferrite left. Of course, with slowly-cooled material the factor remains about 0.9. The grain-size depends mainly on the section, that is, the amount of reduction in the rolls, rather than the finishing-temperature.

In examining a rail-section, it will be found that the grain-size varies considerably. The center of the head will be comparatively coarse, the web much finer, while the edge of the flange is comparatively fine and often shows marked lamination of the ferrite (Fig. 10). Macroscopic etching of a cross-section will show up any segregation that may be present. Fig. 11 shows a section of a rail-head which failed in service. It has been etched with iodine solution and the dark spots show segregation of C and also of MnS. Under the microscope, these dark areas are quite different from the normal section. A longitudinal section shows this well: the steel appears quite streaky, each streak running many inches, sometimes feet. MnS globules in elongated form, also occasionally thin streaks of slag or oxide, usually form the core of such a streak.

The question is frequently raised, whether the old rails were not better than those made to-day; whether our metallurgical progress has not sacrificed quality for tonnage. Undoubtedly, rail-failures are far too frequent; and, in looking for the cause, a careful study was made of a number of old rails, as well as new, to see if the microstructure would tell anything. The grain-size showed but little difference, the size of section having apparently controlled the grain. Macroscopic etching, while showing some of the old rails to be absolutely free from segregation, detected extreme segregation in others—quite as bad as that found in present-day rails. Under the microscope, however, there is a marked difference, due to the carbon-content; in other words, the old rails all had a large excess of ferrite. Their specifications called for from 0.35 to 0.45 per cent. of C. This gave a large excess of ferrite, which shows up distinctly laminated in longitudinal sections (Figs. 8 and 10). Locally-segregated areas, as in Fig. 11, showed an increase in the amount of pearlite, but in the worst cases there was still a comparatively large excess of ferrite, which seemed to minimize the effects of MnS, "ghosts," etc. The modern rails, with carbon from 0.5 to 0.65 and even higher, showed the same segregated areas as in Fig. 11. Under the microscope, how-

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ever, these areas no longer show the excess of ferrite, and the same amounts of MnS, oxide or slag, "ghosts," etc., are now the causes of extreme brittleness.

In regard to piping and consequent segregation, the 0.35 to 0.45 per cent. carbon rail would show at base of head and top of web a maximum segregation of, say, 0.65 per cent. of C. The modern, heavier rail, with from 0.5 to 0.65 per cent. of C and more than 1 per cent. of Mn, frequently shows segregation up to far above 1 per cent. of C, and under the microscope are seen envelopes and plates of cementite, Fe_3C, in and around the pearlite grains, causing extreme hardness and brittleness. Fig. 16 closely resembles such extreme segregation, the thin white envelopes and needles being cementite set in a ground-mass of pearlite. Split heads in rails are mainly due to segregation, occurring in rails from the top of the ingot chiefly. Dudley has pointed out the wedge-action of the hard spots, MnS, slag, etc., which finally causes the head to split vertically. Wickhorst has pointed out the flow of metal on the surface of the head, as shown in Fig. 11, which soon tends to tear apart the head, and in segregated rails a vertical crack occurs in the harder and more brittle interior.

Surface-cracks, checks, and seams in the head are generally found by the inspector. They are filled with oxide, from a mere film to a thick mass, and this oxide is surrounded by an envelope of ferrite, due to decarburization. A crack which has occurred in service may also be filled with oxide, but in this case signs of decarburization are missing, and there is no relation between the grains and the fissure filled with oxide, or, rather, rust.

Basal seams are not so easy to see on inspection, but are readily revealed by macroscopic etching. Their chief position is under the web, and they are found in sections showing slag or oxide and marked decarburization. Such a seam is shown in Fig. 19, a section parallel to the rolling. The large amount of ferrite shows abnormal decarburization. Now this is the point which receives the most strain when a rail is loose, and, as a line of weakness, it rapidly starts to open up and the flange ultimately breaks.

The best method of examination for rails is to etch a cross-section macroscopically and then make longitudinal sections

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through the dark or segregated areas. It has been found by several that MnS alone is not nearly so indicative of brittleness as when accompanied by the thin threads of slag, which generally means the proximity of segregation and the piped area of the ingot.

Tire-Steel.—As usually made, a steel tire shows a fairly-coarse grain, approximately of the size of those in Fig. 22, with thin envelopes of ferrite round the pearlite in steel with 0.6 per cent. of C; while with 0.75 per cent. of C and 0.7 per cent. of Mn the ferrite envelopes disappear. Properly-refined steel shows a very fine grain, even smaller than that in Fig. 21. A Krupp tire generally has a grain of half that size. Some of the Schoen tires showed an extremely fine grain with a sorbitic appearance, closely resembling material quenched from the critical point and reheated to, say, 500° C.

Two kinds of failures have been examined, breakage and "shelling out." In the former, the steel seemed perfectly good, grain-size normal, no marked segregation; and therefore it was concluded that here was another case of "fatigue," that the steel had been subjected to strain at this particular point sufficient to produce progressive fracture. As in the case of axles, so here no trace of any "recrystallization" was met with.

"Shelling out" seems to be caused in three ways: (1) surface-flaws with oxide, similar to Fig. 19; (2) a decarburized surface, which is soft and soon flows, develops cracks which fill with rust until pieces flake off, as shown in Fig. 18, which is a section of a shelled-out tire at the surface; (3) cracks filled with oxide (rust). The first two occur in the process of manufacture; the last occurs in service with apparently good steel. A section will show a certain amount of flow of metal, with cracks set at a slight angle to the surface. These are filled with from films to thick masses of oxide—not slag—due to rusting. There is no decarburization around them, as is usual in the case of slag. It would seem that their cause is similar to that assigned by Wickhorst to certain split-heads in rails. Burning of the steel has often been mentioned as the cause of this kind of failure. A piece of tire was burnt purposely and its structure is shown in Fig. 20. The thin white envelopes to the grains are ferrite, the black masses are oxide. Such a structure has not been met with in tire failures, hence it seem reasonable to assume that burning is not the cause of "shelling out."

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V. *High-Carbon Steel.*

As already remarked, steel with C, 0.55, and Mn, 1 per cent., as rolled, consists of pearlite. $Ac_{1,2,3}$ occurs at 710° to 715° C.; $Ar_{1,2,3}$ at 670° to 655° C.

Similarly, steel with C, 0.9, and Mn, 0.2 per cent. (crucible) is all pearlite, and $Ac_{1,2,3}$ occurs at 730° to 735° C., and $Ar_{1,2,3}$ at 710° to 700° C. The differences in temperature are evidently due to the manganese.

Slow cooling past $Ar_{1,2,3}$ produces well-laminated pearlite, while rapid cooling of small sections, as in air, yields a sorbitic structure.

Overheating occurs at a comparatively low temperature. A bar $\frac{5}{16}$ in. sq., containing C, 0.7, and Mn, 0.068 per cent., was overheated at 800° C., while one containing C, 1.04, and Mn, 0.12 per cent., was overheated at 855° C.

At 1 per cent. of C, are found traces of cementite as thin envelopes to the grains. These envelopes increase with the carbon-content until at 2 per cent. of C they are extremely coarse. Fig. 21 shows such cementite envelopes in a steel with 1.72 per cent. of C.

The boundary-line between cast-iron and steel is unsettled. According to the generally-accepted definition, steel can be rolled out, but cast-iron cannot. From the iron-carbon diagram, the boundary-line occurs at 2 per cent. of C (or 1.7 per cent. according to Ruff), where the eutectic of austenite and cementite makes its appearance. Now, an ingot with a small amount of eutectic can be rolled out, for this cementite is rolled out and broken up just as slag is drawn out in cold-worked wrought-iron. Material with as much as 2.4 per cent. of C has been rolled. Hence, if the presence of the eutectic denotes cast-iron, cast-irons very low in carbon can be rolled and are malleable.

The heat-treatment of high-carbon steels is not simple. With material of not more than 0.9 per cent. of C, heating to just above the line *GOS* causes refining; above that temperature, the higher the temperature of the heating, the coarser the grain. In steels with over 0.9 per cent. of C, the diagram would seem to require that they be heated to just above the line *Sa* to dissolve the cementite envelopes in the austenite, and therefore to refine. In practice, heating to just above the line *PSK* refines the steel (at all events, in small sections), because, on

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heating, the cementite tends to draw itself up into globules and refines itself.

With high carbon (1.6 per cent. and over), heating to higher temperatures, especially in small sections or cold-rolled material, causes the cementite to break up, and graphite is precipitated. Still higher temperatures cause all of the carbon to be taken into solution, and on cooling we have a very coarse grain with cementite envelopes and plates. Fig. 15 shows the corner of a small bar heated to 1,070° C. and slowly cooled. The carbon was 2 per cent. The outside is decarburized, and at the extreme corner we have a little ferrite, due to this cause. The interior, shown on the right, consists of globular cementite set in pearlite. Fig. 16 shows the same steel heated to 1,200° C. and slowly cooled. The ferrite is more pronounced than before, while the cementite now appears as envelopes around and in very coarse grains of pearlite. In the interior, much graphite has precipitated; and Fig. 17 shows the junction between the two areas: that on the right presenting cementite; that on the left, graphite. Fig. 21 shows a steel $\frac{5}{16}$ in. square, with 1.6 per cent. of C, heated to 1,200° C. and slowly cooled. Coarse envelopes of cementite surround the pearlite grains and the whole is very brittle.

Hardening is heating to above the line *SK* and suddenly cooling, as by quenching. The higher the temperature above *SK* the coarser the grain, and therefore the more brittle the material. Theoretically, if cooled quickly enough it should prevent any change in the solid solution austenite. Practically, this is impossible, and quenched material always shows more or less decomposition. Now this decomposition takes place in two ways. The outside of the grain of austenite may decompose into a dark-etching constituent, troostite; the nearer the outside of the bar, the less troostite. Fig. 22 shows a steel with 1.7 per cent. of C heated to white heat and quenched in water. The section is near the outside, and only a small amount of troostite appears. Fig. 23 shows a steel with 1.01 per cent. of C quenched from a yellow heat, section near the center of a bar $\frac{1}{8}$ in. square. The amount of troostite is now very great; only a small amount of austenite remains. The quenching-temperature being lower, the grain is proportionately smaller. The second method of decomposition occurs within the grain itself. Curious chevrons and laths appear, with some semblance of

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octahedral symmetry. This structure is called martensite. Fig. 24 shows it in a steel with 2 per cent. of C, $\frac{5}{16}$ in. square, quenched from 1,100° C. The thin white envelope to the very coarse grain is cementite. Between typical austenite with troostite, and martensite there is a whole range, depending on temperature of quenching, rate of quenching, and size of the section quenched. The grain-size depends on the quenching-temperature.

Tempering is reheating to from 200° to 300° C. to cause a further decomposition of the austenite, whereby brittleness is somewhat overcome, with some loss of hardness. Tempering does not alter the grain-size, but alters the structure of the grain. The martensitic structure becomes more and more pronounced and in time the ground-mass has changed from austenite into troostite and sorbite, and finally sorbite and pearlite are obtained, and then pearlite alone when the material has become completely softened. In order to follow the progress of tempering a small bar may be quenched at a bright yellow heat to get a coarse grain. One end is then heated to a dull red and the other end kept cold. On polishing this section the various changes can be traced from one end to the other. The maximum etching effect occurs at the part heated to about 400° C. According to Heyn, this osmonditic state should divide the stages troostite and sorbite, and coincides with the change in a number of properties.

VI. *Cementation and Case-Hardening.*

Under the microscope the process of case-hardening can be readily followed. It is best to examine the specimens without quenching, for cutting the specimen is much easier and the changes in structure are more easily recognized. On the outside is cementite plus pearlite, or pearlite alone; next ferrite makes its appearance in the pearlite, which gradually decreases in amount until finally the central zone is reached, where there is only the amount of pearlite in the original. Fig. 15 is the reverse of a case-hardened specimen, the position of the ferrite and cementite being reversed. If the material has been quenched it can be let down by heating to, say, 400° C. and cooling in air, when it may readily be cut with the saw. This will not affect the grain-size, and the core can thus be examined. A comparison between fracture and microstructure should always be made.

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VII. *Alloy-Steels.*

The structure of alloy-steels forms a subject by itself. Alloy-steels have been classified into three groups, according to the predominant constituent: (1) pearlitic, (2) martensitic, and (3) austenitic. This, of course, refers to air-cooled specimens.

As an example of the first class is 3.5-per cent. nickel-steel, which closely resembles plain carbon-steel in structure. The grain-size is usually finer, and when quenched the martensite is very much stronger. Other alloy-steels, such as chrome-, nickel-chrome, chrome-vanadium, nickel-chrome-vanadium, silico-maganese, etc., all show pearlite. When quenched and tempered, however, there are profound differences both in properties and structure, martensite being predominant.

Manganese-steel, slowly cooled, shows a typical martensitic structure (Fig. 24), with more or less carbide around and between the grains. Quenching from 1,050° C. produces plain austenite, in polygonal grains. (See Fig. 13.)

Air-hardening and high-speed steels are martensitic, but the structure is by no means typical martensite. Small white grains of carbide are generally found in a complex ground-mass, which at times resembles austenite, at others, martensite. The difference between properly hardened and improperly hardened steel can be recognized readily by those who have studied these steels with some care.

In conclusion, the correct application of metallography in practice ought to be emphasized. It does not compete with chemical analysis or physical testing; it supplements them. It tells about grain-size, segregation, inclusion of impurities, etc., and so is an excellent method for contrasting material and also for tracing the cause of failures. Some of its problems are of easy solution, as, for example, the distinction between properly and improperly refined castings, between wrought-iron and low-carbon steel, good and bad rail-steel, etc. Others, such as the study of hardened alloy-steels, especially high-speed steels, are difficult in the extreme, and require so much patience and skill that many of the workers along these lines have given up the pursuit after realizing its difficulties. As in all scientific work, however, application is rewarded sooner or later, and such subjects as the failure of steel wire in rope, at present unsolved, may in time be unraveled by metallographic methods.

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The Influence of Divorcing Annealing on the Mechanical Properties of Low-Carbon Steel.*

BY HENRY M. HOWE AND ARTHUR G. LEVY, NEW YORK, N. Y.

(Cleveland Meeting, October, 1912.)

THE purpose of the investigation on which this paper is based is to determine whether the structural change which occurs in the slow cooling of steel below the transformation range has an important effect on its engineering properties. We recognize two distinct things which occur in steel when it is brought from above to below the transformation range, first, the transformation proper, that is, the breaking up of the austenite or solid solution of the cementite in the remainder of the iron, and the liberation of its components as undissolved cementite and as ferrite or alpha iron; and second, a structural change consisting in the gradual coalescence of the ferrite and cementite respectively into larger and larger separate masses. Let us distinguish these from each other by speaking of them as "the transformation" and "the structural change" respectively. If the cooling is at all rapid the unevenness of contraction induces severe stress; so that in a sense a slow cooling has a third influence, the prevention of such stress.

This transformation from austenite to ferrite plus cementite can complete itself only below the transformation range; and, because this structural change is a change in the structural arrangement of the products of that transformation, it too can complete itself only at temperatures below that range, say below 725°; and of these temperatures it is only those between 600° and 725° that give the metal sufficient mobility to permit this structural rearrangement to progress with any rapidity.

The transformation is now known to be rapid, but the structural change is relatively slow. The transformation, the liberation of the dissolved cementite from the state of solution in the austenite to a free state, leaves the steel structurally in the

* A contribution from the Metallurgical Laboratories of Columbia University.

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condition of sorbite, a microscopically irresoluble emulsion of the resultant ferrite and cementite. If the further coalescence of that ferrite and cementite occurs as the steel is cooling down from the transformation range, it yields lamellar pearlite, in which the ferrite and cementite are interstratified in the common eutectiform way. If it is brought about by reheating steel which has been brought to the cold with the transformation incomplete, *i. e.*, in the austenitic or martensitic state, the cementite balls up into little pellets in a matrix of ferrite, and the mass is called "granular pearlite." A somewhat like granulation may result from the coalescence of the cementite of lamellar pearlite, as it divorces itself from the pearlitic ferrite. Or, as in steel No. IV. on which we have experimented, the cementite may form reefs or atolls surrounding islands of ferrite, and these masses of cementite would no doubt in time coalesce further and further, so that, if time enough were allowed, the steel would at last consist of one continuous mass of ferrite and one continuous mass of cementite.

Though industrial annealings cannot be made long enough to permit even an approach to this final condition, yet they do permit the coalescence of the initially emulsified ferrite and cementite not only to efface the lamellar pearlite structure completely, and to replace it by a granular or atollic structure, but to allow the coalescence of these granules or atolls to proceed much further.

The Mechanical Properties.—The hardness and the tensile properties, tenacity, elastic limit, elongation, and contraction of area, the properties by which the engineer habitually gauges the fitness of his materials, may conveniently be called the "mechanical properties."

The Question Before Us.—We have long known that the transformation from austenite through martensite to ferrite plus cementite is accompanied by a great gain of ductility and a great loss of hardness. With this loss of hardness, which is only a symptom of loss of cohesion, naturally goes a loss of tenacity and a lowering of the yield point, though in the higher carbon steel this loss is often masked by the simultaneous rise of the tenacity and yield point due to the removal of serious internal stress. The question before us is "How does the structural change affect these mechanical properties?"

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Definitions.—Because the essence of this structural change is the coalescence of the ferrite and the cementite respectively into larger and larger particles, we might indeed call an annealing which brings this about a "coalescing" annealing. But this term might well be thought applicable to sojourns above the transformation range, because these cause the austenite above that range to coalesce into larger and larger grains, and on cooling thence these yield larger and larger masses of the pro-eutectoid element, be it ferrite or be it cementite, and of pearlite. Thus "coalescing annealing" can be restricted to annealing below the transformation range only by an arbitrary convention, and this restriction would be very likely to be unrecognized by many readers. Hence we reject it.

"Granulating" annealing may suffice for an annealing which leads as far as the granular state, but it cannot be used generically to include coalescences which go beyond this stage, nor for those which yield atolls as distinguished from granulation.

"Divorcing" annealing is the best generic name of which we have been able to think for this annealing below the transformation range, causing the coalescence of the ferrite and cementite severally. It is appropriate because in one view the essence of this coalescence is the separation or divorce of the pearlitic cementite and ferrite, which initially are intermingled with the highest degree of intimacy. It is true that when cementite atolls have once been formed the divorce seems pretty complete. Yet the further coalescence of these cementite masses into greater ones may be regarded as a further withdrawal of that cementite from its mate, and thus "divorcing annealing" may without serious stretch be applied to prolongations of this coalescence beyond the point at which the structure has become distinctly atollic and has ceased to suggest pearlite. The name is sufficiently self-explanatory to prevent likelihood of mistaking its meaning.

It may at first seem rather a stretch to speak of the change from sorbite to pearlite as a divorcing, and therefore an annealing which carries the structural change only as far as the stage of pearlite may be spoken of specifically as a "pearlitizing" annealing. But rightly considered this change from sorbite to pearlite is only an early step in a continuous process,

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the separation of the nascent initially ultra-microscopic mixture of ferrite and cementite, through the stages of lamellar pearlite, of the parting of the constituents of pearlite, of their early balling up in microscopic particles, of their later balling farther into macroscopic particles as in the Oriental damaskeened sword blades, to the final state in which all the cementite is gathered into one mass and all the ferrite into another.

We may discriminate readily between these different stages of the transformation and the structural changes below the transformation range by using the following names.

A "tempering" is primarily such a reheating of hardened and hence austenitic or martensitic steel as allows the transformation to proceed to a predetermined degree, without marked structural rearrangement.

A "sorbitizing annealing" is one which, going somewhat farther, allows the transformation to complete itself, but arrests the structural change before the resultant ferrite and cementite have coalesced into microscopically resoluble masses.

A "pearlitizing" annealing is one which permits the structural change to go so far that the ferrite and cementite have coalesced into the form of lamellar pearlite in cooling down out of the transformation range, or into granular pearlite in heating up after a rapid cooling which has arrested the coalescence at an earlier stage.

A "balling up annealing" is one that permits the pearlitic cementite to separate itself from the pearlitic ferrite, and thus to ball up into particles of progressively increasing coarseness.

"Divorcing annealing" may well be used generically to include any or all of these stages. Indeed, because the pearlitizing stage is so very short, *i. e.*, because pearlite forms so very quickly, and its divorce begins so soon, all heatings of any important length in the "divorcing range," that is at temperatures shortly below the transformation range, would naturally be classed together as "divorcing annealings."

Rapidity of the Transformation.—The transformation has been shown by Benedicks[1] to be far more rapid than most of us had believed. Thus he found that the density and electric resistance of a 1.00 per cent. carbon steel, when quenched in water

[1] *Journal of the Iron and Steel Institute*, vol. lxxvii. (1908, No. II.), p. 221.

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with the extreme rapidity of the water quenching mitigated only moderately by wrapping wire about the specimen, were identical with those of the same steel when "annealed." This specimen was quenched from 725°, and occupied 9.5 sec. in cooling from 700° to 100°, or about one-half more than was occupied by a naked specimen quenched in water.

This exact identity as regards density seems to have been fortuitous; for Langley[2] found that variations in the quenching temperature of a like steel even below the transformation range influenced the density materially. Nevertheless these results of Benedicks raise an extremely strong presumption that the transformation had time to complete itself almost perfectly even in his relatively rapid cooling.

Early Observations of the Divorce of Pearlite.—A micrograph by Arnold[3] in 1898 showed that the pearlite of blister steel had been divorced to a marked extent during the very slow cooling which follows the cementation process.

In 1899 Stead[4] reported the occurrence of free cementite in a steel of 0.04 per cent. of carbon, and that "thousands of tons of the thin soft-iron sheets which were annually made had practically all got carbon massed together in them in the state of cementite." Arnold[5] in 1901 recorded cases of more or less advanced divorce of the pearlitic ferrite and cementite into separate masses of cementite and of ferrite, of which the latter is absorbed by the pro-eutectoid ferrite in hypo-eutectoid steels. Such divorce he noted in steels of 0.16, 0.37, 0.50, 0.60, and 0.72 per cent. of carbon. In the 0.72 per cent. carbon steel the divorce seems to have been complete, for he says, "Pearlite as such practically absent. Carbon distributed through the ferrite in rounded globules of Fe_3C." This divorce resulted from the prolonged heating and slow cooling of steel castings, including a sojourn at 950° for about 70 hr., and a cooling which occupied about 6.25 hr. in passing from 700° to 600°.

Though it is now clear to us that the free cementite thus

[2] Langley, *Tool Steel* (pamphlet), p. 9; Metcalf, *The Treatment of Steel* (Crescent Steel Works), pp. 37 to 42 (1884); Howe, *The Metallurgy of Steel*, p. 257 (1890).

[3] *Journal of the Iron and Steel Institute*, vol. liv. (1898, No. II.), p. 185, Pl. XIV.

[4] *Fifth Report Alloys Research Committee, Proceedings of the Institution of Mechanical Engineers*, 1899, pp. 73, 74.

[5] *Journal of the Iron and Steel Institute*, vol. lix. (1901, No. I.), pp. 180 to 182, and Plates X. to XIV., and *The Metallographist*, vol. v., No. 1., p. 2 (Jan., 1902).

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recorded must have come from the divorce of pearlite, yet at that time this may not have been understood. However that may be, the first enunciation of this divorce seems to have been made by Stead[6] in 1902, who reported that it occurred between 620° and 670°.

This divorce was brought into great prominence by Lange[7] in 1903. He reported that a long exposure to an unindicated high temperature, presumably just below the transformation range, induced complete divorce of the pearlite, with the formation of spheroids of cementite and the absorption of the pearlitic ferrite by the free or pre-eutectoid ferrite.

Stead[8] followed this paper almost immediately with an important discussion of the subject, showing in particular that other eutectics had this divorcing tendency; that divorce was most rapid at temperatures shortly below A_1; and that it softened the steel and lowered its elastic limit, and hence should be allowed only in cases in which great softness was desired and the elastic limit could be sacrificed without harm. Hence if a high elastic limit was sought, this divorcing range should be hastened through by a relatively rapid cooling, or in some cases even by quenching, to be followed by an un-divorcing annealing, say at 600°.

One of us had noted independently the divorcing of the copper-silver eutectic in experiments made before this.[9]

Belaiew[10] in 1911 discussed Damascus steel very interestingly, showing that it consists of divorced ferrite and cementite in macroscopically visible masses, and reminding one of us that in 1900 Prof. Tschernoff pointed out to him that it really consists of granular pearlite resoluble by the naked eye. This damask structure is induced by prolonged exposure to a temperature not above redness. Indeed, Belaiew informs us credibly that great care is needed in forging damaskeened steel, for if it is heated above A_1, say 725°, the structure is lost on the recombination of the divorced ferrite and cementite to form austenite.

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[6] *Journal of the Iron and Steel Institute*, vol. lxi. (1902, No. I.), p. 172.

[7] *The Metallographist*, vol. vi., No. 1, p. 9 (Jan., 1903).

[8] The Segregatory and Migratory Habit of Solids. *Journal of the Society of Chemical Industry*, vol. xxii., No. 6, p. 343 (Mar. 31, 1903).

[9] Howe, *Iron, Steel, and Other Alloys*, p. 41 (June 10, 1903).

[10] *Metallurgie*, vol. viii., No. 15, p. 449; No. 16, p. 493; No. 22, p. 699 (Aug. 8, Aug. 22, Nov. 22, 1911).

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It is of great interest that in 1822, as Belaiew informs us, Director Breant of the Paris Mint inferred from his investigation into the damask structure that, in the slow cooling needed for inducing it, the two components of this steel, one rich in carbon and the other poor in carbon and referred to as "iron," separated from each other to form independent masses. Here, save for lacking the names "ferrite" and "cementite," he in effect understood and described the divorce of pearlite into separate masses of ferrite and of cementite.

Influence of Divorcing Annealing on the Mechanical Properties. —On this head we find little recorded information beyond Stead's generalizations already cited, to the effect that this divorce lowers the hardness and the elastic limit.

Arnold's experiments were on steel castings evidently heavily dosed with aluminum, and thus not readily comparable with castings as usually made without this heavy dosing. If we compare those of his castings for which pearlite divorce is recorded with the others, we hesitate to ascribe any special influence to this divorce.

Lange gives the results of only a single test, and in language which suggests that it is a picked one. It is as follows:

Composition.	Per Cent.	Mechanical Properties.	
Carbon	0.38	Tenacity	66,300 lb. per sq. in.
Silicon	0.25	Elastic limit	33,950 lb. per sq. in.
Manganese	0.70	Elongation in 2 in.	38 per cent.
Phosphorus	0.05	Reduction of area	55 per cent.
Sulphur	0.03		

The tenacity and elastic limit are low and the elongation high for a steel casting of this carbon-content. His picture showing the angle bent through by his specimens without breaking indicates that they are very ductile, though some allowance should be made for the probable external decarburization in his probably long exposure, and it is possible that the very great elongation which he records may be due in part to this same cause.

Belaiew quotes from Ischevsky the tests of a tire steel in which divorcing had gone so far as to break up all the pearlite, except a little which it had brought to the granular pearlite stage. The results are as follows:[11]

[11] *Metallurgie*, vol. viii., No. 22, p. 701 (Nov. 22, 1911).

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Composition.	Per Cent.	Mechanical Properties.	
Carbon	0.37	Tenacity	83,200 lb. per sq. in.
Silicon	0.38	Elongation	21.25 per cent.
Manganese	0.98		
Phosphorus	0.05		
Sulphur	0.03		

Lacking information as to the measured length in which the elongation was measured, these data teach little. He further refers to results by Sawin indicating that the onyx type of damaskeening gives a very high elastic limit. Unfortunately, these results are in Russian, and therefore at the moment inaccessible. It is to be hoped that Captain Belaiew may favor us with them.

From a review of the evidence Belaiew infers that the influence of divorcing annealing is to give great malleableness, extraordinary elasticity, and important toughness and ductility, basing his inference in part on language used by one of us.[12]

To sum this evidence up, it tends to show that the effect of a divorcing annealing is to soften (Stead), and to increase the ductility (Lange, Belaiew); but as to the effect on the elastic limit it is contradictory, in part indicating that this annealing lowers the elastic limit (Stead, Lange), and in part indicating the opposite (Belaiew, Sawin).

Finding the evidence so scanty and contradictory, we offer the following results in part to throw light on this question and in part in the hope of drawing the fire of others who have data confirming or opposing our own.

The Present Experiments.—Keeping all other conditions constant, the length of stay in the divorcing range was varied either by varying the rate of cooling from about 900° (quenching in iced brine, in tepid water, in machine-oil, etc.) or by holding at constant temperature in the divorcing range for periods varying from zero to 22.75 hr., and then cooling slowly.

The experiments represented in Fig. 1 and Table I., made in 1896, consisted in cooling like $\frac{3}{8}$-in. square bars of steel No. 22 of 0.21 per cent. of carbon and 1.19 of manganese at different rates from about 900° C., and determining the tensile properties.

[12] Howe, *Iron, Steel, and Other Alloys*, p. 43 (1903).

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TABLE I.—*Influence of the Rate of Cooling from* 900° *on the Mechanical Properties of Steel No.* 22, 0.21 *Per Cent. of Carbon;* 1.19 *Per Cent. of Manganese.*

No. of Bar.	Cooled in	Maximum Temperature.	Cooled to 450° C. in		Ultimate Tensile Strength.	Elastic Limit.	Elongation in 2 In.	Reduction of Area.
		° C.	Hr.	Min.	Lb. Per. Sq. In.	Lb. Per Sq. In.	Per Cent.	Per Cent.
38	Iced brine at 6° C...	880	.....	.....	228,920	.....	2.0	1.30
36	Water at 14.75° C...	850	.....	.....	219,100	.....	2.0	2.67
37	Water at 15.25° C...	840	.....	.....	213,330	.....	1.0	0.67
	Average.....	.....	.....	.....	216,210	.....	1.5	1.67
42	Machine oil.....	895	.....	.....	174,910	.....	2.8	1.75
43	Machine oil.....	890	.....	.....	173,450	.....	3.0	1.06
	Average.....	.....	.....	.....	174,180	.....	2.9	1.4
34	Blast of cool air.....	890	.....	.....	88,387	54,629	29.0	57.18
35	Blast of cool air.....	890	.....	.....	87,406	Below 56,665	27.6	57.74
	Average.....	.....	.....	.....	87,890	55,650	28.3	57.46
44	Air of room.....	875	.....	.....	86,827	53,649	27.0	58.79
45	Air of room.....	875	.....	.....	85,769	52,147	29.0	59.52
31	Air of room.....	875	.....	.....	86,956	.....	28.05	55.74
32	Air of room.....	875	.....	.....	87,637	57,232	27.0	57.27
	Average.....	.....	.....	.....	86,800	54,340	27.76	57.83
27	Air in iron pan.....	875	0	3	87,706	54,192	26.9	57.10
28	Air in iron pan	875	0	3	87,087	54,519	27.3	57.80
	Average.....	.....	.....	.....	87,400	54,355	27.1	57.45
40	Hot fireclay.....	875	0	30	83,812	Below 51,753	31.0	59.07
41	Hot fireclay.....	875	0	30	82,927	48,699	31.0	58.95
	Average.....	.....	.....	.....	83,370	50,226	31.0	59.01
30	Furnace.....	890	4	8	79,576	.....	29.1	54.52
33	Furnace.....	890	4	8	80,630	44,221	27.2	54.98
	Average.....	.....	.....	.....	80,100	44,221	28.1	54.75

Those represented in Figs. 2, 3, and 8, and in Table II. consisted in cooling like $\frac{5}{8}$-in. round bars of steel No. IV. of 0.21 per cent. of carbon and 0.05 of manganese from 900° under varying conditions, and determining their tensile properties, hardness, and microstructure. Certain bars were air-cooled directly from 900°; the rest were furnace-cooled to the upper part of the divorcing range, 685°. From this point on some were air- and some furnace-cooled immediately on reaching 685° (ordinates *B* and *C* of Figs. 2 and 3); some were held for variable periods at 685° and then air-cooled (ordinates *D* and *E*); and some were thus held at 685° and then furnace-cooled to 330° (ordinate *F*), whence they were air-cooled. All the heatings were in an atmosphere rich in carbonic oxide, to restrain decarburization. After these heat treatments the outside of the specimens was turned off to remove any decarbur-

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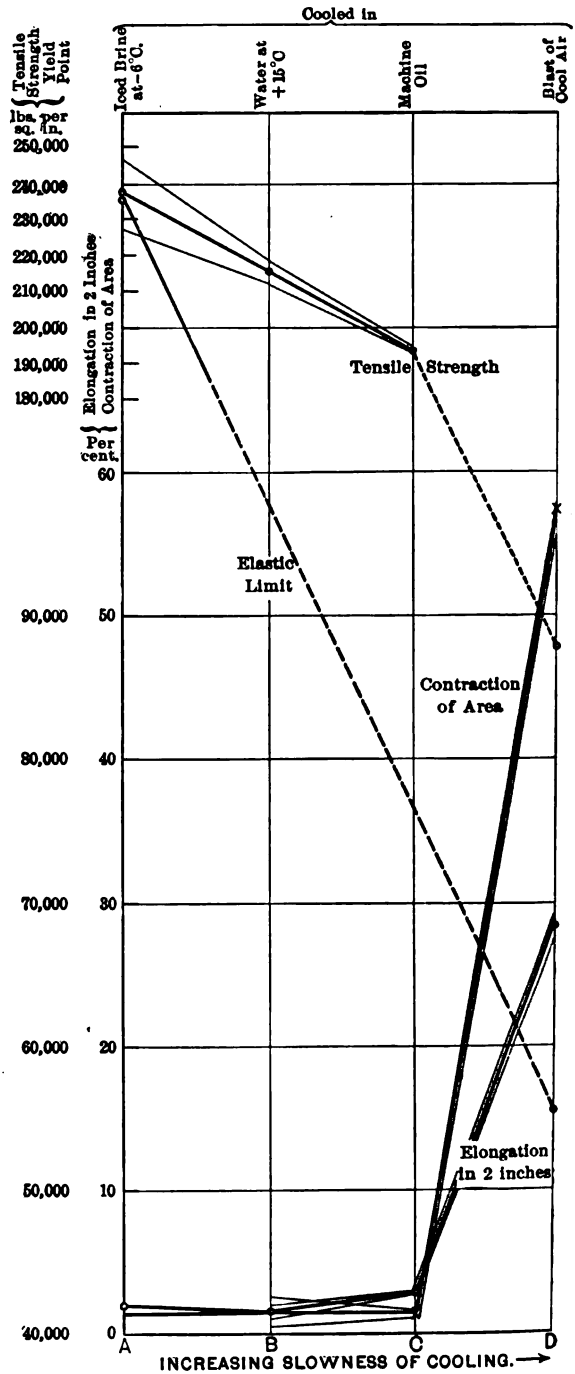

NOTE TO FIG. 1.—The vertical scale from 40,000 to 60,000 lb. is the

FIG. 1.—INFLUENCE OF THE RATE OF COOLING FROM 900° C. ON THE 1.19 PER CENT.

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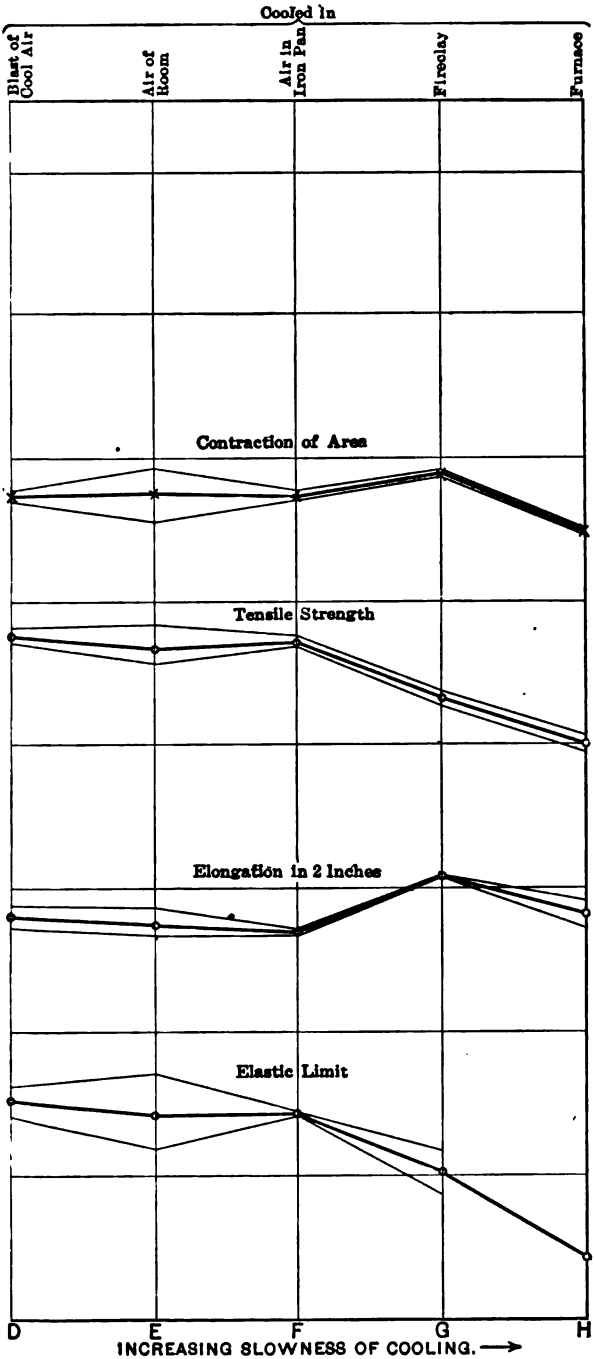
The several curves represent the maximum, minimum, and average



same as that of Figs. 2 and 3. From 60,000 to 250,000 lb. it is smaller.

MECHANICAL PROPERTIES OF STEEL No. 22, 0.21 PER CENT. OF CARBON, OF MANGANESE.

of the several properties after the various treatments.

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ized layer, and the specimens were then reheated to 300° to crystallize any metal made amorphous by the machining. The experiments are described in detail in the appendix. The "hardness" results given in Table II. were reached by means of the Brinell drop-test, measuring the area of the impression caused by the impact of a ram falling from a fixed height.

TABLE II.—*Influence of Prolonging the Stay in the Divorcing Range on the Mechanical Properties of Steel No. IV.*, 0.21 *Per Cent. of Carbon*, 0.05 *Per Cent. of Manganese.* (*See Figs.* 2 *and* 3.)

All Heated to 900°.

No. of Bar.	Group No.	How Cooled. From 900° to 680° ± 5° In	How Cooled. Held at 680° ± 5° to 710° For	How Cooled. From 680° to 710° to Room Temperature In	Ultimate Tensile Strength.	Yield Point.	Elongation in 4 In.	Reduction of Area.	Hardness Brinell Drop.	Micrograph Figure
			Hrs.		Lb. Per Sq. In.	Lb. Per Sq. In.	Per Cent.	Per Cent.		
61	1	Air	Nil.	Air	56,370	40,610	21.25	66.08	20.8	A.
64	1	Air	Nil.	Air	58,080	41,600	20.25	64.08	.....	.....
73	3	Air	Nil.	Air	64,240	47,986	19.5	60.54	.....	.....
76	3	Air	Nil.	Air	64,535	47,036	21.0	61.95	.....	.....
		Average	.....	.....	60,800	44,300	20.5	63.16	.....	.....
62	1	Furnace to 685°..	Nil.	Air	57,870	37,480	17.75	58.46	19.3	B.
63	1	Furnace to 685°...	Nil.	Air	58,500	37,800	18.50	53.00	.....	.....
69	2	Furnace to 680°...	Nil.	Air	51,825	35,611	18.25	56.37	.....	.....
70	2	Furnace to 680°...	Nil.	Air	57,412	38,150	17.60*	55.52	.....	.....
		Average	.....	.....	57,150	37,260	18.02	55.84	.....	.....
71	2	Furnace	Nil.	Furnace to 300°...	53,833	36,008	22.25	57.66	.....	.....
72	2	Furnace	Nil.	Furnace to 300°...	55,317	33,858	18.80*	58.46	.....	.....
		Average	.....	.....	54,570	34,930	20.52	58.06	.....	.....
66	1	Furnace to 685°...	1.5	Air	55,270	35,970	19.25	57.57	19.3	D.
68	1	Furnace to 685°..	1.5	Air	55,360	34,840	20.50	58.96	.....	.....
74	3	Furnace to 675°..	1.5	Air	55,860	34,974	22.0	57.09	.....	.....
		Average	.....	.....	55,500	35,260	20.58	57.87	.....	.....
65	1	Furnace to 685°..	23.75	Air	59,400	40,510	18.50	57.54	18.6	E.
67	1	Furnace to 685°..	23.75	Air	57,060	38,270	21.00	59.61	.....	.....
80	3	Furnace to 675°...	22.5	Air	56,285	35,660	21.2	60.61	.....	.....
		Average	.....	.....	57,580	38,150	20.23	59.25	.....	.....
75	3	Furnace to 675°..	22.5	Furnace to 330°...	56,055	37,108	22.2	60.45	.....	.....
77	3	Furnace to 675°..	22.5	Furnace to 330°...	56,310	33,558	21.0	59.94	.....	.....
78	3	Furnace to 675°..	22.5	Furnace to 330°...	56,620	35,750	22.0	60.19	.....	.....
79	3	Furnace to 675°..	22.5	Furnace to 330°...	55,550	35,846	21.0	60.58	.....	.....
		Average	.....	.....	56,134	35,565	21.55	60.29	.....	.....

* Elongation measured in 5 inches.

The Results.

Successive Stages in the Prolongation of the Exposure to the Divorcing Range.—The results which we offer represent exposures to the divorcing range of temperature, varying in length from the extreme brevity of a quenching in iced brine, ordinate *A* of Fig. 1, to the 23.75 and 22.5 hr. holdings in the upper part of that range, about 685°, of ordinates *E* and *F* of Fig. 2.

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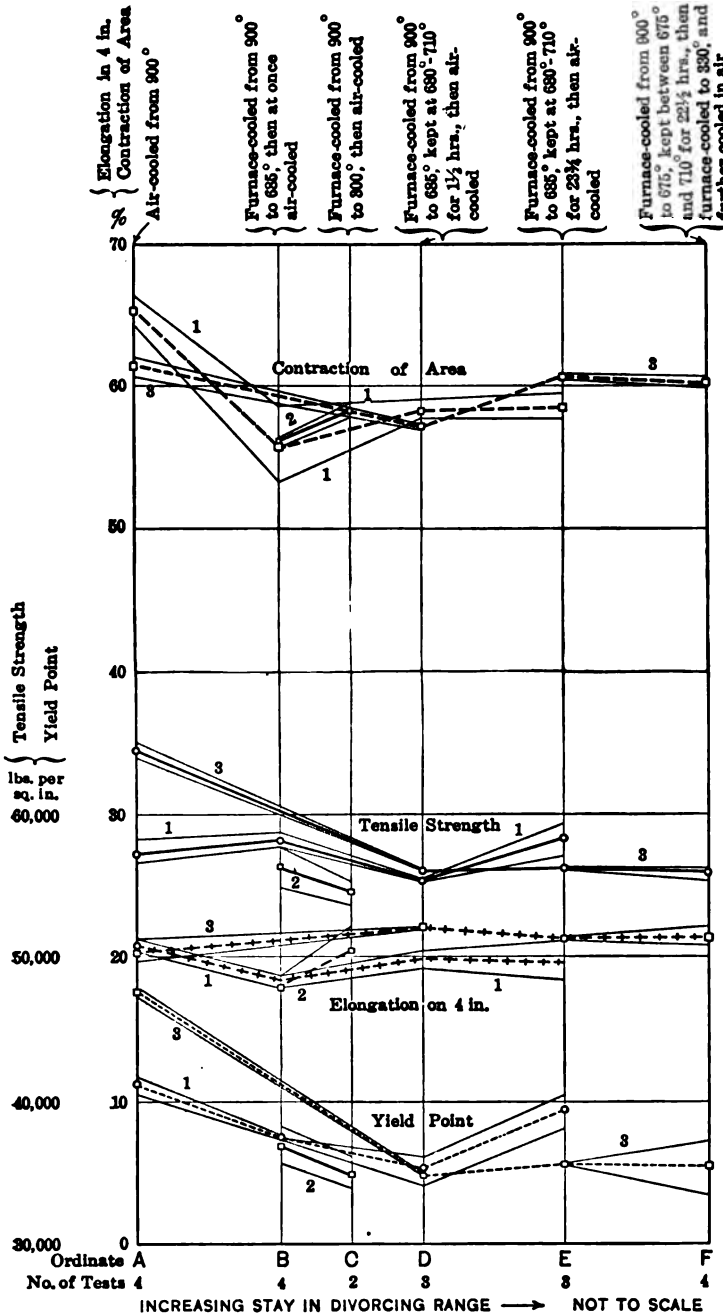

FIG. 2.—INFLUENCE OF PROLONGING THE STAY IN THE DIVORCING RANGE ON THE MECHANICAL PROPERTIES OF STEEL No. IV., 0.21 PER CENT. OF CARBON, 0.05 PER CENT. OF MANGANESE.

The several curves represent the maximum, minimum, and average of the several properties for each group of specimens after the various treatments.

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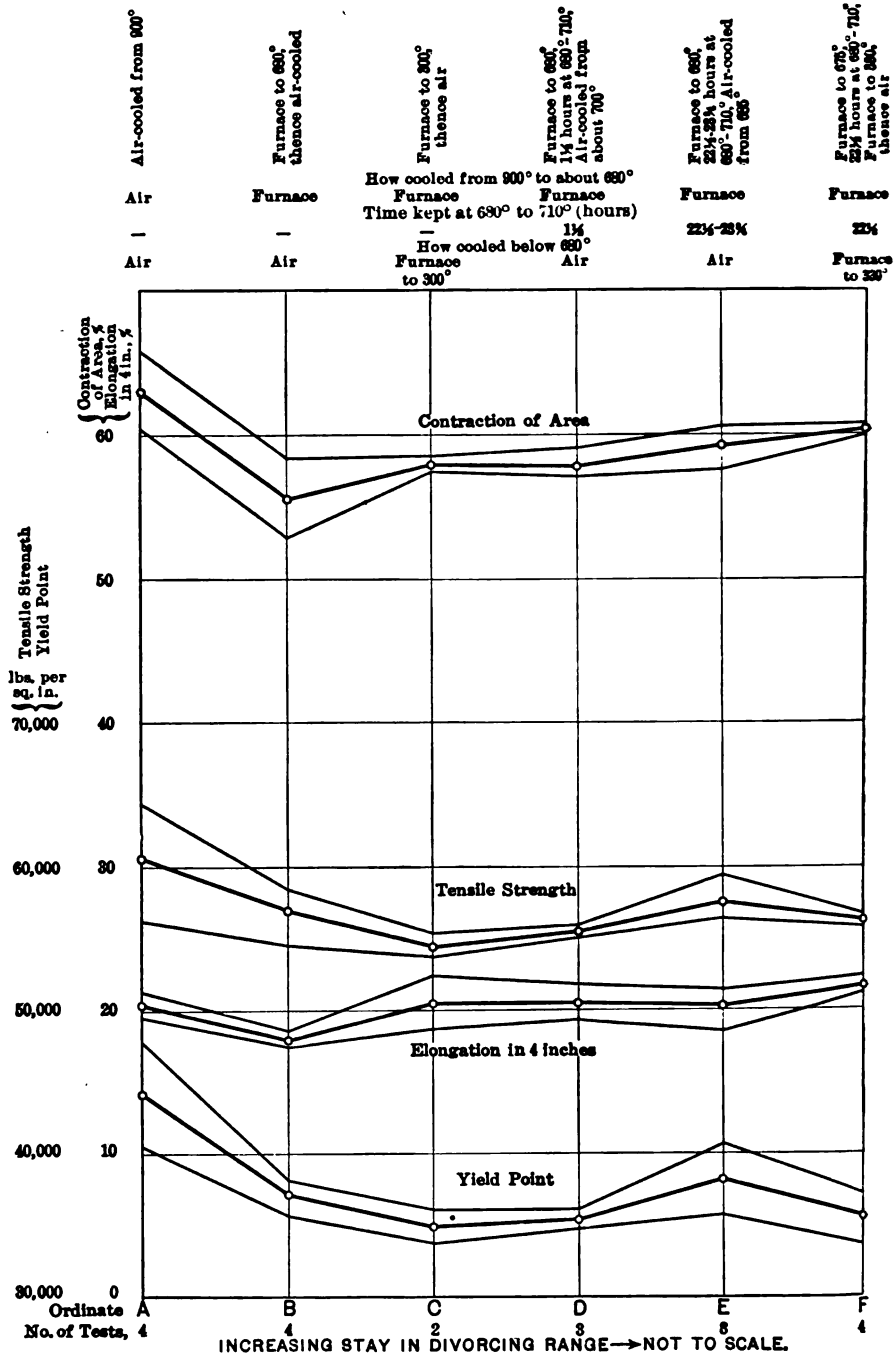

FIG. 3.—INFLUENCE OF PROLONGING THE STAY IN THE DIVORCING RANGE ON THE MECHANICAL PROPERTIES OF STEEL NO. IV., 0.21 PER CENT. OF CARBON, 0.05 PER CENT. OF MANGANESE. CONDENSED FROM FIG. 2.

The several curves represent the maximum, minimum, and average of the several properties of all the groups collectively after the various treatments.

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For convenience we may divide these successive prolongations of this exposure into

1, from that of iced-brine quenching to that of an accelerated air-cooling;

2, from that of an accelerated to that of a retarded air-cooling;

3, from that of air-cooling to that of a simple furnace-cooling to 685°, followed by a 1.5 hr. stay at 685°; ordinate *D* of Figs. 2 and 3; and

4, further prolongations, ordinates *E* and *F* of Figs. 2 and 3.

Of these, the first prolongation should, in view of Benedicks's results, be taken as representing the completion of the transformation, and the further prolongations as representing the structural change, from sorbite through pearlite to further and further divorce of that pearlite.

First Prolongation from Iced-Brine Quenching to Air-Cooling.—The results shown on ordinates *A* to *D* of Fig. 1 bring out plainly the already known great decrease of the yield point and tenacity and increase of ductility which accompany the transformation in steel of this carbon-content. The tenacity and yield point are here decreased by 63 and 76 per cent. respectively, and the elongation and contraction of area increased 13 fold and 43 fold respectively.

Prolongation 2, from Accelerated to Retarded Air-Cooling.—Fig. 1 shows at a glance that neither this nor the further prolongations have any influence at all comparable with that of the first prolongation, from the almost instantaneous passage of the iced-brine quenching to that of an accelerated air-cooling, ordinate *D* of Fig. 1. This special prolongation 2 is shown by the horizontality of the curves from ordinate *D* to ordinate *F*, Fig. 1, to be without important influence, and this evidence is supported by the slightness of the influence of the further prolongation from ordinate *F* to ordinate *G* of this figure. Figs. 2 and 3 throw no light on the influence of this specific prolongation, because they do not represent any cooling intermediate between an unaccelerated air-cooling and a furnace-cooling. It is true that, though the individual members of Groups 1 and 3 on ordinate *A* of Fig. 2 agree closely among themselves, these two groups do not agree. We defer discussing this disagreement till we shall have investigated it.

Prolongation 3, from Air-Cooling to Furnace-Cooling to 685°,

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Followed by 1.5 Hours at 685°, ordinates *A* to *D* of Figs. 2 and 3.—In view of the fact that in every case there is a material fall of the yield point, and that the amount of fall, though varying somewhat from group to group, is closely concordant among the members of each group, we can hardly escape the conviction that a true and habitual effect of this prolongation is to lower this point. This tends to confirm Dr. Stead's remark that divorce lowers the elastic limit. The tensile strength also falls in all three groups, though by a very small amount except in Group 3. On the other hand, the changes in ductility are but little more than may well arise in heat treatments intended to be identical. There is a like fall of tenacity and yield point in the most nearly corresponding prolongation in the cooling of Steel No. 22, as shown between ordinates *F* to *H* of Fig. 1.

Further Prolongation, 4, Ordinates D to F of Figs. 2 and 3.—The prolongation of the stay at 685°–710° from 1.5 to 23.75 hr. has no important effect on the tensile properties, beyond perhaps increasing the ductility slightly, but it causes a slight further softening. Thus the loss of elastic limit asserted by Dr. Stead seems to come to an end after a stay of moderate length in the divorcing range, and not to continue during this prolongation 4.

Slow Temperature-Fall as Such vs. Exposure to Divorcing Range Temperature.—It remains to test the truth of the assumption on which we proceeded in the experiments represented by ordinates *B*, *D*, and *E*, Figs. 2 and 3, that the influence of a slow as opposed to an air-cooling from Ar_1 down is due, not to its being a cooling as such, but solely to its involving an exposure to temperatures which, while below Ar_1, so that the whole of the metal quickly becomes ferrite and cementite instead of austenite or martensite, are yet high enough to give the mobility needed for the divorce of the pearlite and the progressive coalescence of each of its parted constituents into larger and larger particles. This involves the further assumption that, once the transformation is complete, a slow cooling acts solely through permitting divorce, and that the long exposure to temperatures below the rapid d vorce range which slow cooling involves is incompetent to induce any special change peculiar to those lower temperatures and unable to occur at the higher temperatures of rapid divorce. Proceeding on this assump-

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tion in the experiments of ordinates *B*, *D*, and *E* of Figs. 2 and 3, we cooled the metal rather quickly, in air, after a sojourn of variable length at a temperature in the upper part of the divorcing range, 685°, on the ground that all that a slow cooling could do would be to carry a little further the divorce and coalescence which has been occurring at 685°, by affording an exposure of appreciable length to temperatures slightly below 685° at which there is still appreciable mobility, and that apart from this increase of the total opportunity for divorce a slow cooling would have no effect.

It is of course understood that a truly rapid cooling, even below the transformation range, may induce severe stress, and thereby influence the properties of the metal taken as a mass; but we do not believe that air-cooling from 685° down can induce any important stress in specimens as small as ours, $\frac{5}{8}$ in. in diameter and as low in carbon.

In order to test this assumption that nothing occurs in a slow cooling below the divorcing range beyond that which would occur if the stay in the divorcing range were prolonged, we tried the experiments represented by ordinates *C* and *F* of Figs. 2 and 3. The conditions of these experiments differ from those of ordinates *B* and *E* solely in following the stay at 685° with a furnace-cooling to about 300° instead of with an air-cooling.

In the former case, *i.e.*, when steel is cooled slowly from 900° to 685°, just below Ar_1, but does not stay at 685°, the substitution of a slow cooling for an immediate air-cooling in effect increases the opportunity for divorce by a large fraction. Thus the stay in the upper part of this divorcing range was only about 10 min. in the case of the specimens represented by ordinate *B* of Fig. 2, air-cooled on reaching 685°, consisting, as this stay did, of the time spent in cooling slowly from, say 700° to 685°. But the specimens represented by ordinate *C*, cooled slowly from 685°, had, in addition to a stay at and above 685° of about this same length, a further exposure of 45 min. in cooling from 685° to 600°, a range of temperature in which the mobility, though continuously decreasing, is yet great enough to permit divorce to proceed at a considerable rate. Therefore it is in accordance with our assumption that the effect of this slowing of the cooling from 685° down has, on each of the five

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properties determined, an influence like in sign to the influence of prolonging the 685° exposure to 1.5 hr. The tenacity and yield point fall slightly, and the elongation and contraction of area increase slightly. This is shown by the direction of the lines of Group 2 in Fig. 2, from ordinate *B* (slow cooling to 685° followed by air-cooling) to ordinate *C* (slow cooling from 900° to 300°), those of tenacity and yield point falling and those of elongation and contraction of area rising.

But after a 22.5 hr. exposure to 685° the substitution of a slow cooling (ordinate *F*) for an air-cooling (ordinate *E*) increases the total opportunity for divorce by only a very small fraction. The fact that this substitution has very little effect on any of the properties tested, the curves being nearly horizontal from *E* to *F*, then, is a strong indication that our assumption now under discussion is true.

Results previously reached by one of us [13] with the steel represented in Fig. 1, agree fully with these. In these older experiments the specimens were cooled slowly in the furnace from about 900°, and this slow cooling was interrupted at various temperatures in and below the transformation range by quenching in cold brine. The decrease of the "missing carbon" [14] continued till the quenching temperature had sunk below 575°; the increase of elongation, and contraction of area continued to below 532°; the loss of hardness to below 512°; and that of tenacity to below 263°. This continuance of these changes to temperatures well below the divorcing range is to be referred to two causes jointly. First, because as with ordinate *B* of Figs. 2 and 3 the furnace-cooling was not followed by any stay in the divorcing range, the exposure to that range was so brief that the successive lowerings of the cooling at first offered an appreciable increase of the oppor-

[13] Howe, *Journal of the Iron and Steel Institute*, vol. xlviii. (1895, No. II.), p. 258.

[14] *Idem*, p. 275. When hardened steel is dissolved in nitric acid for the Eggertz colorimetric method for carbon, part of the carbon dissolved in the austenite, or martensite, *i.e.*, of the hardening carbon, is volatilized; this method, therefore, finds less carbon than in the same steel when annealed. The difference we may call the "missing carbon." While the total quantity of dissolved or hardening carbon is probably much greater than the missing carbon, so that it would be incorrect to report the missing carbon as hardening carbon, yet as the missing carbon increases in rough proportion with the hardening carbon, its variations furnish an indication of those of the latter.

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tunity for divorce. Second, the gradual lowering of the quenching temperature resulted in less and less violent internal stress, and thus progressively increased the softness and ductility, and lessened the tenacity. The persistence of the decrease of missing carbon to below 575° is to be interpreted as representing the continued coalescence of the cementite into masses which, because of their size, the better resisted the tendency to volatilization.

That appreciable stress should be induced by quenchings even from these low temperatures is to be inferred from Langley's [15] finding that the density of steel was slightly lowered by a water-quenching even from a temperature as low as 100° C.

A practical application of this principle is that, if a divorcing annealing is to be given for softening the metal or for other purpose, it is more economical to hold the temperature in the upper part of the divorcing range, say from 650° to 700°, and cool thence as rapidly as is convenient, and compatible with the avoidance of considerable internal stress, thus leaving the furnace free for subsequent lots, rather than to hold for a shorter time in this range, and then to cool slowly in the furnace. For during that slow cooling the divorce is progressively retarded by the decreasing mobility of the metal, so that the whole time of such slow cooling is used to poor advantage, and the latter part of it, after the temperature has sunk to say 400°, is practically wasted.

The Structural Change Shown Micrographically.

Progress of the Pearlite Divorce.—The air-cooling from 900°, Fig. 4, *A*, leaves the product of the transformation of the eutectoid austenite, *i.e.*, of the "hardenite," in the state of ill-defined pearlite, a somewhat sorbitic pearlite, with the movement towards divorce arrested at an early stage.

In Fig. 4, *B*, representing furnace-cooling to below the transformation range, 685°, followed by air-cooling, we see black zebra striped islands of pearlite, together with some black-bordered white cementite islands. The black border seems to be a shadow effect. In Fig. 5, in which the cementite has been blackened by etching with sodium picrate, the black-

[15] *The Treatment of Steel* (Crescent Steel Works), p. 42 (1884). Howe, *The Metallurgy of Steel*, p. 31 (1890).

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ness of these islands, and the habitual absence of any border from them shows that they are cementite, in some cases with slight pearlite inclusions.

In Fig. 4, *D*, representing furnace-cooling to 685° followed by 1.5 hr. at 700°, the structural divorce has gone further, and in Fig. 4, *E*, it has gone much further, and has now reached the atollic stage, reefs of black-rimmed white cementite enclosing large lagoons of ferrite, readily recognized by the characteristic etching figures.

These micrographs, *B*, *D*, and *E*, selected as typical, must be taken with some caution, because at each of the stages here represented there occur, in other parts of the same section, both wholly undivorced pearlite, and well divorced atolls; and though the undivorced pearlite becomes more rare, and the cementite atolls more frequent as the sojourn at 685° increases, yet this change is not so marked as one would expect.

The presence of some atolls even as early as in the specimen shown in Fig. 4, *B*, air-cooled as soon as 685° was reached, and the retention of some undivorced pearlite in the specimen shown in Fig. 4, *E*, even after a 22.75 hr. sojourn at 685°, indicates strongly that the pearlite masses in the same section differ greatly among themselves in their attachment, some parting company at the first cross word, others bearing and forbearing patiently.

Philosophy of the Divorcing Annealing—The reason why even a very protracted divorcing annealing has so little influence on the mechanical properties may be that the changes in these mechanical properties are only the resultant of the changes in the properties of the ferrite, and of the pearlite respectively. The divorce of the pearlite, the coalescence of its parted constituents into larger and larger masses, and the grain-growth of the pro-eutectoid ferrite, are going on side by side. One of these may tend to change the mechanical properties in one direction, the other in the opposite direction. Each of these component changes in the mechanical properties may be very considerable, yet their resultant may habitually be small. Under special conditions which favor one of these changes more strongly than the others the resultant may change apparently capriciously; so that apparently unimportant varia-

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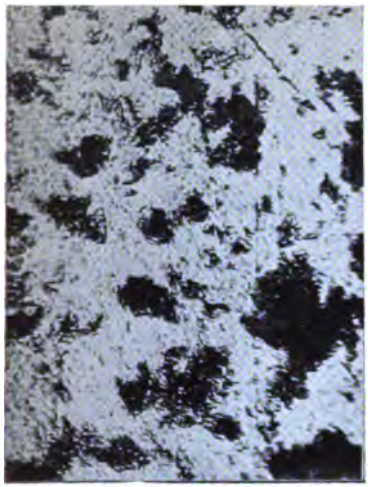
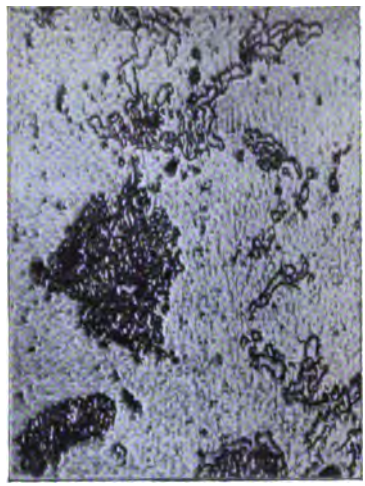
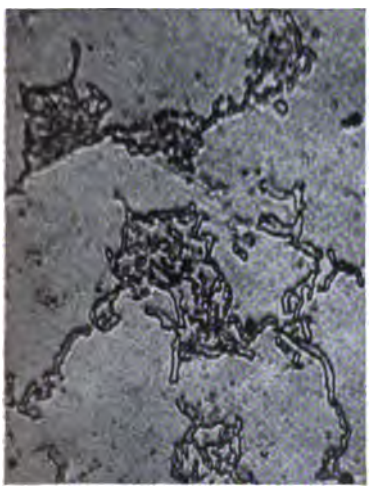


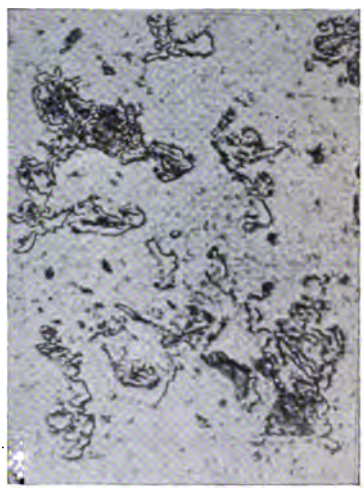
A, Air-cooled from 900°. Sorbitic pearlite.



B, Furnace-cooled to 685°, then air-cooled. Incipient atolling.



D, Furnace-cooled to 685°, held 1.5 hr., then air-cooled. Atolling well advanced.



E, Furnace-cooled to 685°, held 23.75 hr., then air-cooled. Atolling further advanced.

FIG. 4.—MICROSTRUCTURE OF STEEL NO. IV., 0.21 PER CENT. OF CARBON, 0.05 PER CENT. OF MANGANESE, HEATED TO 900°, AND COOLED IN VARIOUS WAYS.

The designation letters A, B, D, and E, refer to the treatments represented by ordinates A, B, D, and E of Figs. 2 and 3.

All × 500. Etched with picric acid.

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tions in the conditions may cause unexpected and at first sight anomalous changes in the mechanical properties.

One consideration we cannot escape, that the influence of these cementite masses is clearly strengthening and embrittling, because the mass as a whole remains stronger, with a higher elastic limit and less ductility than those of a lower carbon steel in which the quantity of cementite is far less or even nil. Hence our natural inference is that the balling up of the cementite as such should tend to increase the ductility, and to lessen the tenacity and the elastic limit, because in whatever way the cementite exerts its strengthening and embrittling effect, we

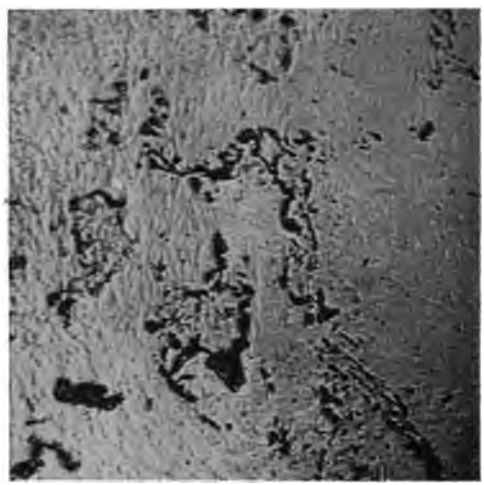

FIG. 5.—MICROSTRUCTURE OF STEEL NO. IV. AFTER TREATMENT E, HELD 23.75 HR. AT 685°. × 500. ETCHED WITH SODIUM PICRATE.

expect it to exert less effect when it is massed up in relatively large particles, leaving the ferrite less supported and freer to behave in its own natural way.

If we start from this premise, then the slightness of the resultant suggests that the grain-growth of the ferrite has the opposite effect, lessening the ductility, and increasing the tenacity.

But the behavior of such a composite mass, first under elastic stress, and later under plastic deformation, depends upon so many considerations that no theory can be accepted with the least confidence, till it has been established experimentally.

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Light could be thrown on this question by noting how the behavior of ultra low-carbon steel, in which the influence of ferrite grain-growth ought to predominate, compares with that of an only slightly hypo-eutectoid steel in which the pearlite divorce and cementite coalescence ought to have the chief influence. A strictly eutectoid steel should be avoided, because it might contain pro-eutectoid cementite locally, with perturbing effects, and also because its divorce would probably be very slow.

(At first sight one might expect to get valuable light on this problem from the fact that, as the carbon-content of steel increases, from specimen to specimen, the tenacity and elastic limit increase and the ductility decreases, whence we are tempted to refer to the presence of the unyielding islands of pearlite a strengthening and embrittling action on the mass as a whole, so that the pearlite not only lessens ductility by interfering with the flow of the ferrite, but increases tenacity by opposing that flow.

But another theory of the increase of tenacity and elastic limit with the carbon-content, long regarded by one of us as a possible one, has just received very important support from Professor Benedicks. That theory is that the embrittling and strengthening effect of increasing carbon-content, even in the slowly cooled state, is due to the retention of allotropic iron, caused by the presence of that carbon. In 1896 one of us[16] pointed out, in connection with Professor Sauveur, that in certain cases the strength of steel which consisted of a mechanical mixture of martensite and ferrite was far greater than could be explained readily by the mechanical supporting action of that martensite, and offered as one of two alternative theories that the ferrite retained some allotropic iron, one reason for that retention being the presence of carbon, the brake action of which he and the other allotropists recognized as a necessary link in the conditions which cause the hardening of steel. Professor Benedicks has now shown[17] first that this allotropic iron is probably gamma, and second, that such retention of an allotropic modification at temperatures far below that at which

[16] Howe and Sauveur, *Journal of the Iron and Steel Institute*, vol. xlix. (1896, No. 1.), p. 178.

[17] On allotropy in general, and that of iron in particular, advance proof from the *Journal of the Iron and Steel Institute*, Sept., 1912.

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the main allotropic change is due, may not only occur but may represent equilibrium and not hysteresis. This suggests that, as the carbon-content of the steel as a whole increases, there increases with it, not by hysteresis but in equilibrium, the proportion of gamma iron which the ferrite retains permanently, and that the strengthening and embrittling effect of increasing carbon-content is due primarily to this progressive change in the constitution of the ferrite. Further, that as gamma iron can dissolve carbon abundantly, so with the increase in the gamma-iron content of the ferrite goes a corresponding increase in its carbon-content. This would modify Professor Benedicks's long-held theory of ferronite, and lead us to believe that the carbon-content which he ascribed to ferronite is really not a fixed but a variable carbon-content of ferrite, increasing progressively with the carbon-content of the steel.)*

Divorcing Annealing of Hyper-Eutectoid Steel.—It would not be safe to infer from these results that Captain Belaiew's inferences as to the effect of prolonged divorcing annealing are wrong, though they disagree radically with our results. We should remember here that we are dealing with a relatively low-carbon steel, and that he has prominently in mind the probably hyper-eutectoid Oriental sword steel. It may well be that a thorough divorcing annealing benefits such steel, if not precisely in the way which he indicates, yet very greatly, and in particular by lessening its brittleness, and raising its elastic limit. For the pro-eutectoid cementite in such a steel plays a part which has no parallel in our low-carbon steel, forming strong sheets and even plates which may well cause great brittleness, and lower the elastic limit very greatly, exerting an influence many fold greater than that of the pearlite divorce and ferrite grain-growth of our low-carbon steel.

The great loss of ductility which the presence of masses of pro-eutectoid cementite may cause is shown in Fig. 6, representing the tensile properties of a steel of 1.14 per cent. of carbon, and 0.24 per cent. of manganese,[18] cooled slowly from a

* The matter in parentheses was not in the paper as presented, but was added on Nov. 8, 1912.

[18] The microstructure of this same steel after these heat treatments is shown by one of us in the *Proceedings of the American Society for Testing Materials*, vol. xi., p. 262 (1911). Micrograph *B*, of Row 2, Pl. II., of that paper shows the strong cementite network in this steel after 6 hr. at 900°. In the specimen held for 10 min. at 900° plates of pro-eutectoid cementite can be seen under higher powers.

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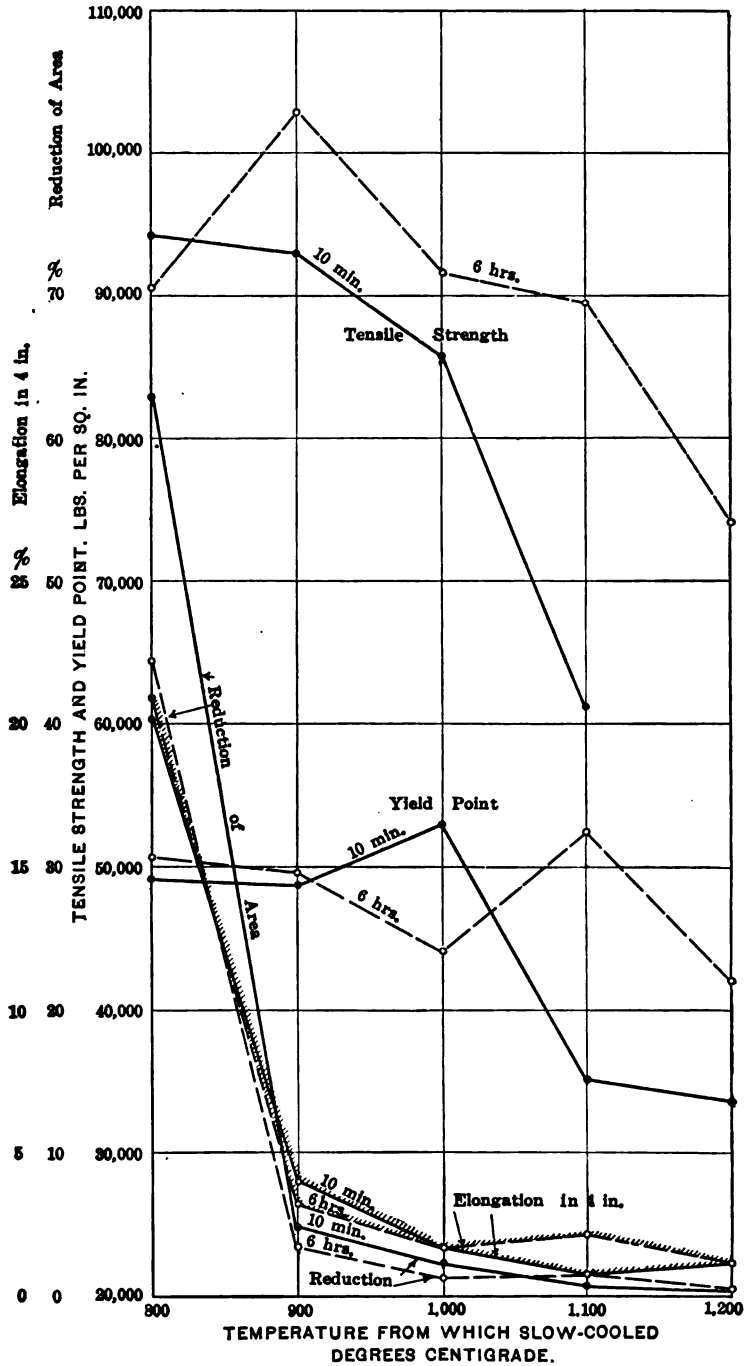

FIG. 6.—INFLUENCE OF HEATING TO VARIOUS TEMPERATURES ON HYPER-EUTECTOID STEEL OF 1.14 PER CENT. OF CARBON WHEN FURNACE-COOLED.

NOTE TO FIG. 6.—The ⅝-in. round bars as received from the maker were heated to 625° for 1.5 hr., cooled slowly in the furnace, then severally reheated to one of the various temperatures shown, varying from 800° to 1,200°, and again furnace-cooled. They were turned down to about 0.45 in. in diameter before tensile testing. The specimen shown on the 6-hr. curve at 1,200° was heated at 1,200° for only 2 hr. Its elongation is measured in 2 in., not in 4 in.

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series of high temperatures, after a stay at those temperatures of 10 min. in some cases and 6 hr. in the others. The good ductility which remained till after the maximum temperature of heating had reached 800° fell abruptly by the time this temperature was raised to 900°, slightly above the top of the transformation range. A slow cooling from 900° should cause sheets of pro-eutectoid cementite to form, and under the microscope they are in fact plainly visible. On the other hand, it is not till the heating temperature rises above 1,000° that an important lowering of the tenacity, and of the yield point occurs.

This embrittling effect of these sharp masses of pro-eutectoid cementite tends to support Captain Belaiew's belief that the extreme stage of divorcing annealing of the Oriental sword blades lessens the brittleness, by balling this cementite up into masses which, though indeed large, are rounded, and therefore not likely to gash the ferrite during deformation, and thus to precipitate rupture.

That this massing of the pro-eutectoid cementite into relatively large masses should thus affect the ductility far more than the elastic limit and tenacity seems but natural. The elongation and contraction of area take place during the plastic deformation which occurs between the yield point and rupture. It is but natural that the large strong masses of pro-eutectoid cementite should, during plastic deformation, be prone to cut gashes too deep to be healed by the pearlitic ferrite, and thus precipitate rupture before plastic deformation has proceeded far, and thus lead to low ductility. On the other hand, one does not see clearly why they should lower the tenacity or the elastic limit greatly; and accordingly we are not surprised to find that no marked lowering of these properties occurs till the heating temperature has risen so high as to exaggerate the coarseness of the pro-eutectoid cementite.

Further Notes on the Behavior of Hyper-Eutectoid Steel.

The data in Fig. 7 show the progressive recovery from the effects of overheating hyper-eutectic steel to 1,100°. In the slow passage down through the transformation range which followed this overheating, a pro-eutectoid cementite network of course formed. In the subsequent reheating, the effect of

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which is given in Fig. 7, this network persisted till the temperature rose into the transformation range, and then gradually melted away during the passage up through that range. But in the following slow cooling new masses of pro-eutectoid cementite formed. If the temperature rose only part way through the transformation range, say to 800°, the initial coarse pro-eutectoid cementite remained in part and prevented any material gain in ductility.

To raise the temperature still higher, say to 900° or 1,000°, would indeed lead to a further reabsorption of the old pro-eutectoid cementite; but this was incompetent to restore the ductility, because of the new pro-eutectoid cementite which formed in the subsequent slow cooling through the transformation range. Hence it is but natural to find that the elongation and contraction of area of overheated hyper-eutectoid steel are raised but little by the heating to 800° or any other temperature.

But as regards tenacity and elastic limit the case stands very differently. Here the beneficial effect of lessening the old coarse network by reheating into the transformation range is not at first offset by the formation of the new pro-eutectoid cementite in cooling down, and hence the tenacity and elastic limit rise to a maximum as the reheating temperature rises to between 800° and 900°, to fall off only slowly with the further coarsening of the new pro-eutectoid cementite with further rise of the heating temperature.

From these data we infer that coarse pro-eutectoid cementite has much less effect on the tenacity and elastic limit than on the ductility, as would naturally be inferred from the considerations given above, that ductility represents capacity to endure plastic deformation, a capacity which would naturally be decreased by the coarseness of the pro-eutectoid cementite, whereas tenacity and yield point represent rather the behavior under elastic deformation, in which the effects of the coarseness of the pro-eutectoid cementite would naturally be much less marked.

And here we have an explanation of the contrast between hypo- and hyper-eutectoid steel as regards grain refining by reheating coarsened metal. Coarsened hypo-eutectoid steel is benefited by heating to above the transformation range, and

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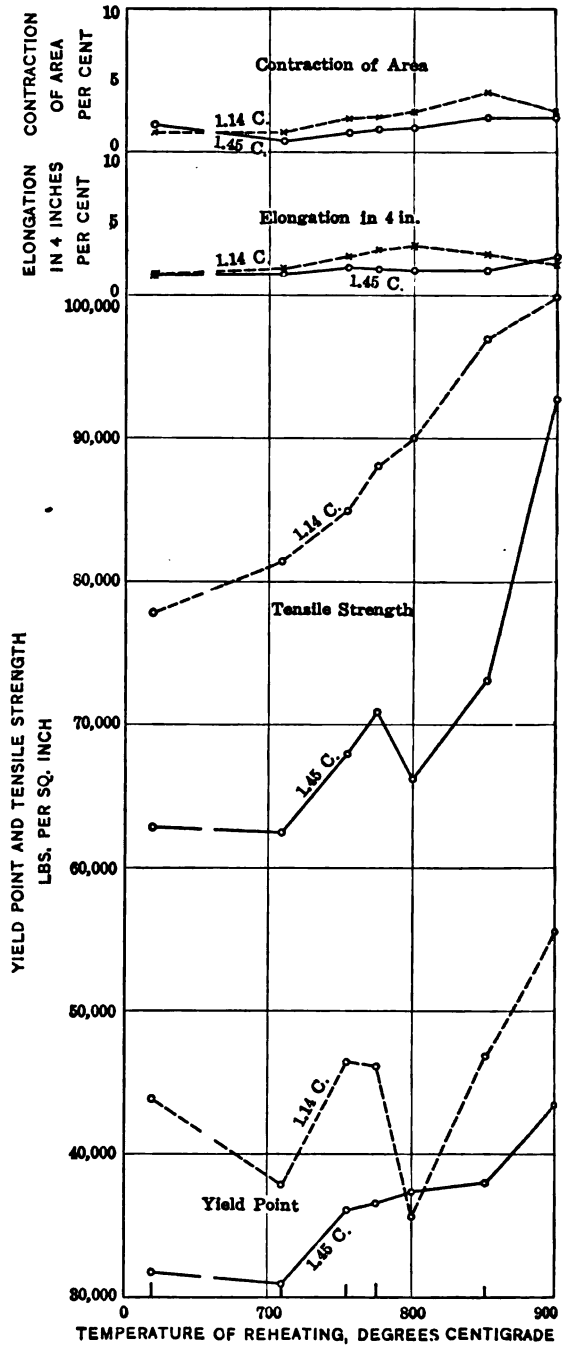

FIG. 7.—THERAPEUTIC INFLUENCE OF HEATING TO VARIOUS TEMPERA-
STEELS (CARBON 1.14 AND 1.45

NOTE TO FIG. 7.—The $\frac{5}{8}$-in. round bars as received from the maker were various temperatures shown, varying from 710° to 1,300°, and again furnace-sile testing. The point at the left-hand end of the curves represents the

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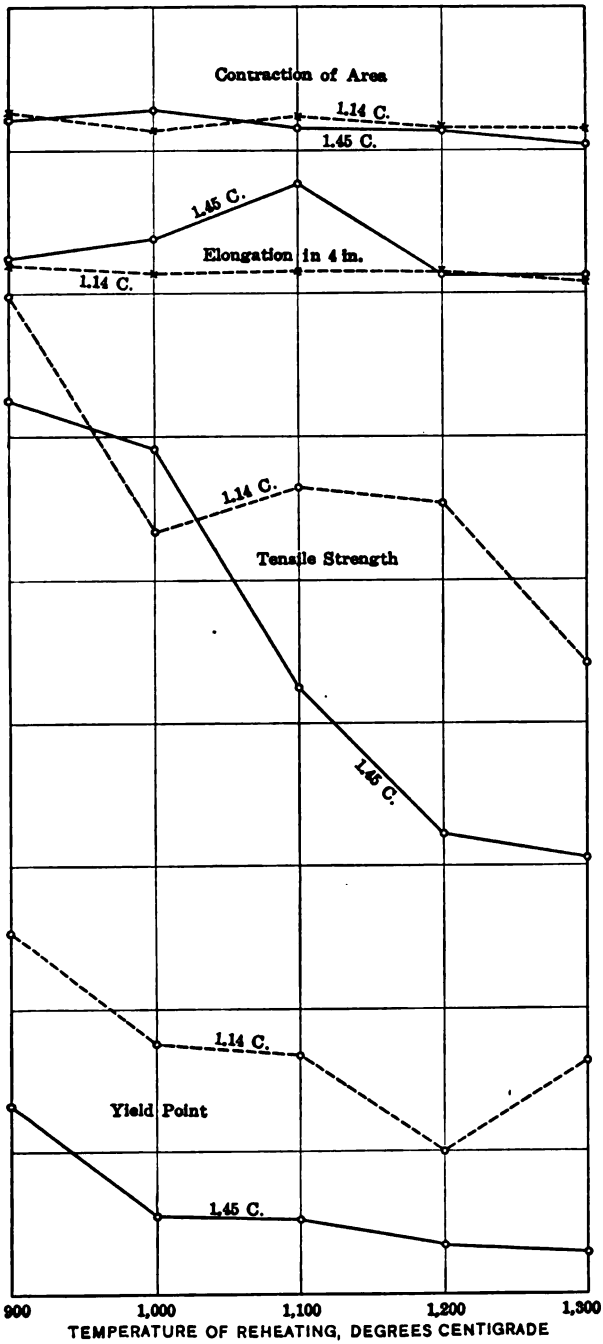

TURES ON PREVIOUSLY PATHOGENICALLY OVERHEATED HYPER-EUTECTOID PER CENT. RESPECTIVELY).

heated to 1,100° for 1 hr., furnace-cooled, reheated severally to one of the cooled. They were turned down to about 0.45 in. in diameter before ten- specimen which underwent only the first of these heatings, to 1,100°.

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thereby wiping out the initially coarse network structure, even if this is followed by a slow cooling, because the new pro-eutectoid ferrite which forms in finer masses on again cooling slowly is much less harmful than the initial coarser network. But hyper-eutectoid steel is not so improved, because any heating which rises far enough into the transformation range to cause enough re-absorption of the pro-eutectoid cementite to remove its embrittling effect is necessarily followed in the subsequent cooling by a re-formation of new masses of pro-eutectoid cementite which are prohibitorily embrittling.

The essence of the difference is that pro-eutectoid ferrite, itself most ductile, is not fatally embrittling when in such masses as it forms in slow cooling through the transformation range, whereas pro-eutectoid cementite is. The embrittling caused by overheating hypo-eutectoid steel is due to the coarseness of heterogeneousness and not to the pro-eutectoid element itself, whereas the pro-eutectoid element of hyper-eutectoid steels is in and by itself extremely embrittling unless very minutely subdivided.

Hence hyper-eutectic steel, if it has been coarsened, is habitually refined rather by forging than by simple heat treatment, by breaking up the new nascent pro-eutectoid cementite by the mechanical distortion of forging in the passage down through the transformation range. If it were to be refined by heat treatment alone, it should be cooled rapidly through the transformation range to restrain the coalescence of the pro-eutectoid cementite, and in the case of thick or very high-carbon pieces, this rapid cooling should be arrested, or at least greatly slackened, when the transformation range has been passed, lest the mass be cracked by the internal stress. We have found that the mechanical properties of coarsened hyper-eutectoid steels are very greatly improved by reheating to above the transformation range, quenching, and again reheating gently to remove the hardening caused by the quenching.

Summary of the Effect of Prolonging the Divorcing Annealing on the Mechanical Properties.

A. Low-Carbon Steel (carbon 0.21).

1. Variations in the length of stay at the temperature of most active divorce, about 685°, even if that stay is prolonged

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to 23.75 hr., have but an insignificant effect compared with the effect of variations in the rate of cooling through the transformation range.

2. The prolongation of this stay caused by substituting a retarded for an accelerated air-cooling has but trifling influence.

3. The prolongation caused by substituting a stay at 685° of 1.5 hr. for a retarded air-cooling lowers the yield point decidedly, and probably lowers the tenacity, but only slightly.

4. The further prolongation of this stay at about 685° from 1.5 to 23.75 hr. has no effect beyond a slight further softening, and perhaps a slight further increase of ductility.

5. Thus the lowering of the elastic limit asserted by Dr. Stead seems to come to an end, or at least to be very greatly retarded, after a relatively brief stay.

6. The influence of a slow cooling as such from the divorcing range is identical in effect on the mechanical properties with a prolongation of the exposure to that range, so that a given degree of divorcing with its results is given more quickly by a stay in the upper part of the divorcing range, followed by a cooling as rapid as is compatible with the avoidance of undue stress, than by a slow cooling from that range down.

B. Hyper-Eutectoid Steel.

7. These results of divorcing annealing on low-carbon steel throw little light on the effect of such an annealing on hyper-eutectoid steel, which, because of its pro-eutectoid cementite, may be expected to follow laws radically different from those which govern hypo-eutectoid steel. But the evidence tends to support the belief that prolonged divorcing annealing may increase the ductility of hyper-eutectoid steel to a degree far beyond that noted with hypo-eutectoid steel.

8. Starting from the normal state, hyper-eutectoid steel loses its ductility very abruptly when heated to temperatures between 800° and 900°, but its elastic limit and tenacity are not greatly lowered till the heating temperature is raised further.

9. If previously coarsened by slow cooling from above the transformation range, reheating and slow cooling do not restore the ductility thus lost. But such coarsened steel does regain a large part of its tenacity and elastic limit on reheating to about 900°. With further rise of the reheating temperature

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the tenacity and elastic limit again fall off, though much less abruptly than the ductility.

Acknowledgments.—The experiments were made in the metallurgical laboratories of Columbia University, in part under a grant from the Carnegie Institution of Washington.

We have to thank Messrs. Wheaton B. Kunhardt and John A. Mathews for the specimens of steel, Prof. L. D. Norsworthy of the Department of Civil Engineering, Columbia University, for the tensile testing of Steel No. IV., Mr. John H. Hall, Metallurgist of the Taylor-Wharton Iron and Steel Co., for the tensile testing of Steels Nos. VIII. and IX., Messrs. Booth, Garrett & Blair for determining the carbon and manganese of Steels Nos. IV., VIII., and IX., and Prof. Wm. Campbell of Columbia University for valued advice.

APPENDIX.

Description of the Experiments.

The Temperatures are given throughout in degrees centigrade.

The Steels Used.—Their composition is as follows:

Table III.—*Compositions of the Steels Used.*

Number.	Represented In	Carbon.	Silicon.	Manganese.	Phosphorus.	Sulphur.
22	Fig. 1 and Table 1.....	0.21	0.31	1.19	.....	.....
IV.	Figs. 2 to 5 and 8 and Table 2.....	0.21	.....	0.05	0.013	0.010
VIII.	Figs. 6 and 7.....	1.14	0.12	0.24	0.016	0.009
IX.	Fig. 7.....	1.45	0.14	0.16	0.009	0.006

Foot Note.—The carbon and manganese of Steels Nos. IV., VIII., and IX., were determined by Messrs. Booth, Garrett & Blair—the other elements by the makers of the steels. The analysis of Steel No. 22 is by Mr. Clement Le Boutillier, late chief chemist of the Taylor-Wharton Iron & Steel Co.

Other History of these Steels.—Steel No. 22 is that on which experiments by one of us are recorded in the *Journal of the Iron and Steel Institute*, vol. xlviii. (1895, No. II.), p. 258 (H. M. Howe), and vol. xlix. (1896, No. I.), p. 170 (Howe and Sauveur).

Steel No. IV. is the gift of Wheaton B. Kunhardt, Treasurer of the Carpenter Steel Co.

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Steels Nos. VIII. and IX. are the gift of Dr. John A. Mathews, Operating Manager of the Halcomb Steel Co. Experiments on these steels are described by one of us in the *Proceedings of the American Society for Testing Materials*, vol. xi., 1911, p. 262 (H. M. Howe). The micrographs in that paper, Rows 1 and 2 of Pl. II. and Row 9 of Pl. IV., are from wads of Steel No. VIII., heated alongside of the specimens of which the tensile tests are represented in Figs. 6 and 7 of this present paper. Other experiments on these steels are described by us in *Internat. Zeits. Metallographie*, 1912, III., p. 4, and *International Association for Testing Materials*, VIth Congress, 1912, II., 4.

Heat Treatment. Steel No. 22.—These specimens were heated in the small tubular gas muffle shown in Figs. 9 and 10, *Trans.*, xxiii., 536 (1893). In this they were raised to about 900° and at once cooled in the various ways indicated. Most of the treatments were applied to two specimens each, and these two specimens were heated side by side.

Steel No. IV.—The specimens were heated in a horizontal steel gas-pipe 2.5 in. in diameter and 30 in. long, capped at each end and enclosed in an alundum tube. This outer tube was heated by means of the resistance of a nichrome wire wound spirally about it. Decarburization was restrained by enriching the atmosphere with carbonic oxide by means of charcoal set within the inner tube. Beyond this, after the heat treatment the specimens were turned down from their initial size of $\frac{5}{8}$ in. diameter to $\frac{1}{2}$ in., so as to remove any decarburized outer layer, and they were then reheated to 300°, so as to recrystallize any steel made vitreous by the machining.

The fact that the tenacity and yield point of Group 3 are practically the same after the 22.5 hr. stay at 685° as after the 1.5 hr. stay there indicates that decarburization was effectively prevented, because if it had taken place to an appreciable degree it would have lowered both these properties. Further evidence in the same direction is the actual increase of the tenacity and yield point in Group 1 between the 1.5 hr. stay and the 23.75 hr. stay at 685°.

The eight specimens of Group 1 were treated simultaneously and side by side, up to the point at which they were severally removed from the furnace for air-cooling. The same is true of the specimens of each of the other groups of Steel No. IV. In

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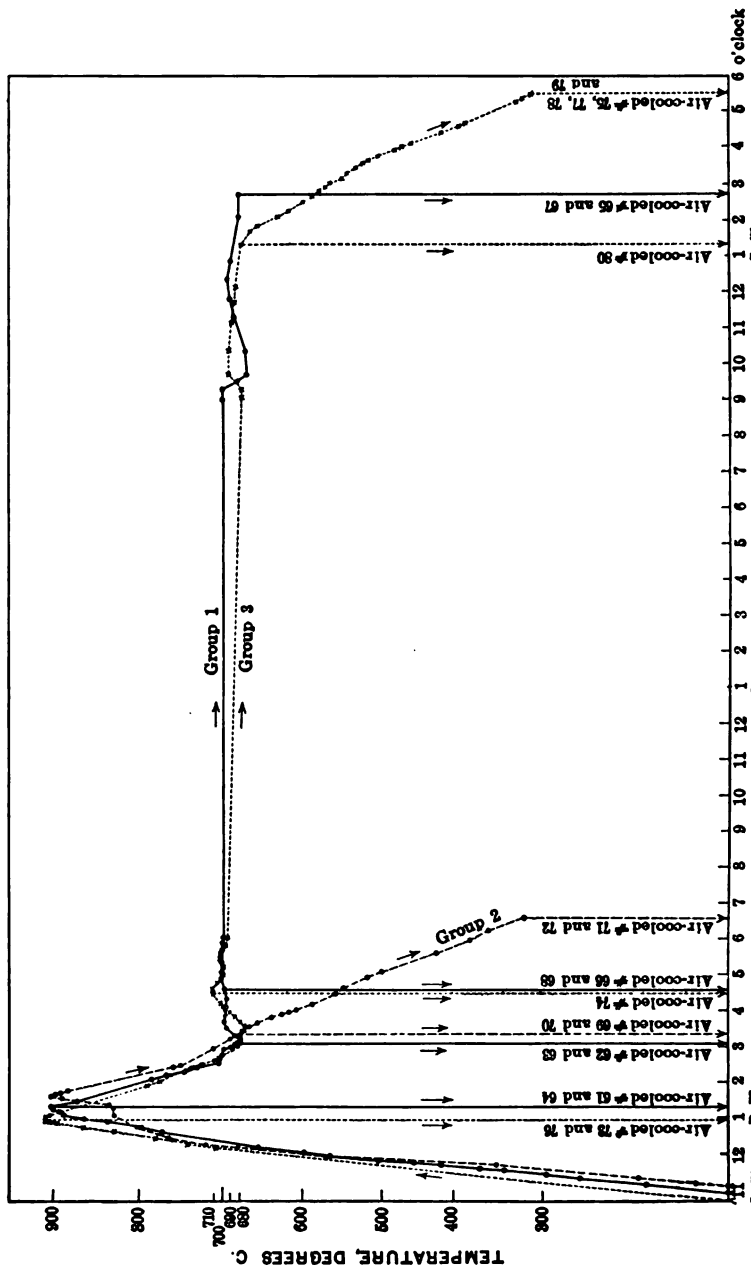

FIG. 8.—THERMAL RECORD OF THE TREATMENT OF STEEL NO. IV.

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the air-cooling the bars were laid horizontally with their ends resting on fire-bricks in the open air of the laboratory, and far enough apart not to influence each other sensibly.

The 23.75 hr. heating of Group 1 was not under direct observation during the night, but previous experience under like conditions makes it reasonably certain that the temperature did not deviate far during this period. The 22.5 hr. heating of Group 3 was controlled during the night by an autographic pyrometer, which showed that the temperature then did not pass beyond the limits of 675° and 710° given for the beginning and end of the night period. Fig. 2 shows all the results, and gives the maximum, minimum, and average for each group, except in a very few cases in which the results agreed so closely that the differences between them cannot conveniently be represented thus. Fig. 3 shows the maximum, minimum, and average of these same results taken together instead of being split up into groups.

The course of the temperature in the heatings and coolings of the three groups of Steel No. IV., is shown in Fig. 8.

Steels Nos. VIII. and IX.—In the experiments represented in Fig. 6, the specimens were heated in an iron gas-pipe capped at each end, set within a closed graphite crucible in a large gas crucible furnace. Here they were heated to 625°, to remove initial stress, etc., held there 1.5 hr., furnace-cooled, reheated in the same position to the temperatures shown, held there for the time shown, and again furnace-cooled.

In the experiments represented in Fig. 7 the treatment was the same, except that the first high-temperature holding was at 1,100° for 1 hr., for the purpose of inducing a coarse network, the cure of which was undertaken in the following heating. The first heating may be called pathogenic, and the second therapeutic.

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Electric Heating and the Removal of Phosphorus from Iron.*

BY ALBERT E. GREENE,† CHICAGO, ILL.

(Cleveland Meeting, October, 1912.)

PROCESSES for the removal of phosphorus from iron or steel are steadily assuming greater importance in view of the abundance of high-phosphorus iron-ore and the diminishing supply of pure ore. In the present methods of removing phosphorus by the basic open-hearth or basic Bessemer process, or even by electric processes, complete control over the metallurgical conditions is either impossible or has not yet been realized. The removal of phosphorus has so long been accomplished along certain well-defined lines, that when the electric furnace made its appearance metallurgists adhered to the same old reactions used in the older processes and failed to appreciate and take advantage of the new forces at their disposal. One object of the present paper is to set forth more clearly the metallurgical reactions by which phosphorus can be removed from iron, and to show the necessity of controlling the conditions, and particularly the temperature, by means other than combustion in the furnace-chamber.

Invariably, the commercial removal of phosphorus has been accomplished by oxidation. The reactions involved in oxidation of phosphorus in general consist, first, of the formation of P_2O_5, and second, the combination of this oxide with lime, forming calcium phosphate, which latter is held in a slag high in iron oxide. In the present-used processes, the phosphorus is never removed without the assistance or presence of considerable quantities of iron oxide. The oxygen for removal of phosphorus in the basic open-hearth process comes largely through the medium of iron oxide and furnace-gases, but not

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from lime, and the lime itself cannot prevent the reduction of phosphorus back into the metal out of slags from which iron oxide is largely reduced; and likewise in the basic Bessemer process, although here the oxygen comes originally from air, yet iron oxide formed in the blow is the essential carrier of oxygen.

Among the more important metallurgical conditions which control the reactions of phosphorus are the temperature, the presence of oxidizing agents or reducing agents, or both, and the intensity of the resulting oxidizing or reducing conditions, or, in other words, the equilibrium conditions governing the reactions involving oxygen; the influence of slags, including the influence of oxides therein having affinity for phosphorus oxide or other phosphorus compounds, the solubility in the slag of such phosphorus compounds, the presence of oxide of like chemical nature to phosphorus oxide and tending to displace it; and the affinity of the reduced metal for phosphorus. In processes in which the heat is supplied by combustion it is not possible to maintain complete control over all the above conditions. In the open-hearth furnace the temperature is maintained by combustion of gas in the furnace-chamber, and the atmosphere is oxidizing to a greater or less extent. Any control over the reducing conditions in an open-hearth furnace must therefore come from reducing agents in the metal, and very limited control, if any, can be had over the action of agents such as silicon or dissolved carbon. Since iron oxide cannot be kept from forming in the slag under influence of the furnace atmosphere, it is apparent that the silicon and carbon in the metal are the agents which must be relied upon to prevent such oxidation.

To hold the strongly-acid oxide of phosphorus in the slag requires a fluxing agent having strong affinity for phosphorus and strongly basic in chemical nature, like lime. Calcium phosphate is not at all difficult to reduce, however, at an elevated temperature and with strong reducing agents, and either silicon or carbon is capable of reducing it. This statement holds true even though the carbon or silicon be dissolved in iron, although the intensity of the reducing action is then less, possibly owing to the "pulling action" of the iron itself on these reducing elements. The temperature may, of course, greatly influence the action on the slag of these reducing

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agents contained in the metal, and may so change their affinity for oxygen as to prevent reduction of certain oxides in the charge, as is the case with carbon and phosphorus. In an open-hearth furnace calcium phosphate will be reduced from the slag back into the metal when the conditions of equilibrium between the iron oxides in the slag and the reducing agents in the metal are such as to reduce the amount of iron oxide below a certain limit. Silicon is such a strong reducing agent that practically always, except in certain cases to be noted later, it must be oxidized out of the metal before the phosphorus is attacked. If present in the bath it reacts with any oxide of phosphorus present and reduces it. The presence of large amounts of silica in the slag or lining of the vessel is ordinarily detrimental to removal of phosphorus.

The influence of carbon, the other important reducing agent in the metal, is largely dependent on the temperature, its affinity for oxygen being less than that of phosphorus at low temperatures, and greater at temperatures above about 1,450° C. Thus, at low temperatures, carbon dissolved in the iron is not a strong enough reducing agent to reduce calcium phosphate held in a slag high in iron oxide, but at higher temperatures it is. Thus, either by means of silicon or by means of carbon contained in the metal, or, of course, by solid carbon, the iron oxide in the slag may be readily reduced, and when such reduction proceeds far enough calcium phosphate is also reduced and the phosphorus goes back into the metal.

The fact that at low temperatures phosphorus is more easy to oxidize out of the metal than carbon is the basis of several modifications of the open-hearth process, such as the Krupp washing-process, the Bertrand-Thiel process, etc. The slight differences in these processes, such as the heating up of ore, lime, and pig-iron together, or the preliminary heating of the ore and lime and subsequent use of molten pig-iron, do not change to any great extent the final result as to the method of removing phosphorus. The manner of carrying out this oxidation of phosphorus determines the degree of activity and frothiness of the slag, but essentially the reaction is the formation of calcium phosphate in the presence of considerable iron oxide.

In the operation of the basic Bessemer process, the chemical

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reactions involved in the separation of phosphorus are in reality practically the same as those in the open-hearth process. In the Bessemer process, temperature is maintained by combustion as in the open-hearth, the combustible materials being the silicon, phosphorus, iron, carbon, etc., of the charge, and also these are burned with an excess of oxygen to produce the necessary heat. Oxidation of the phosphorus results from oxygen of air primarily, but iron oxide formed by contact with air acts as a carrier of oxygen, just as in the open-hearth process the slag serves as a carrier of oxygen from the furnace-gases.

A large proportion of the heat evolved in the basic Bessemer process results from the oxidation of phosphorus, but it has been found that a certain proportion of silicon is necessary in the pig-iron to produce the requisite final temperature. Silicon is oxidized first, together with iron, and thereby the temperature of the bath is raised above that known as the critical temperature between carbon and phosphorus, and above this critical temperature the phosphorus is not removed appreciably until after both the silicon and the carbon are largely oxidized out. In certain cases, with low silicon, oxidation of phosphorus takes place from the early part of the blow simultaneously with that of carbon. This is what would be expected if the temperature be held at about the critical point, since at this temperature the oxidation of carbon and the formation of calcium phosphate take place with equal readiness.

An interesting example of the effect of metallic reducing agents in connection with the basic Bessemer process is the rephosphorization of the metal on the addition of spiegel. The spiegel enters the metal at a high temperature and through a slag containing considerable quantities of calcium phosphate. At an elevated temperature, both silicon and manganese are easier to oxidize than phosphorus and these two elements tend to reduce the phosphorus from the slag back into the metal. From my recent experiments, it has been found that at a low temperature phosphorus, on the contrary, is more easily oxidizable than manganese in the presence of lime and can be separated as calcium phosphate without oxidation of manganese. The reactions involved will be discussed more fully later.

On comparison of the conditions in the Bessemer and the

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open-hearth process, it is seen that the controlling factors in the separation of phosphorus are almost identical. Lime is, of course, necessary to hold the phosphorus in the slag. Combustion is the source of heat and iron oxide is always present and usually in large amounts; silicon oxidizes before phosphorus, and the oxidation of carbon before or after phosphorus is determined by the temperature. Lack of complete control over the reactions resulting from producing the requisite temperature by combustion in the furnace-chamber, and necessitating loss of iron as oxide in the phosphate slag, is the very feature which electric heating can correct.

The electric furnace has been found well adapted to removing phosphorus by oxidation in the old way, but not commercially, and on careful consideration it is difficult to see how it could do so commercially in competition with the open-hearth. The process of removal of phosphorus in the Heroult furnace, for example, as practiced by the United States Steel Corporation, has been to charge blown steel into the furnace and melt on top of it a slag of lime and iron oxide. The reactions are identical with those in a basic open-hearth. And in spite of the fact that the temperature can be maintained high, the removal of phosphorus in a large furnace takes anywhere from 20 min. to 1.5 hr., according to the amount of phosphorus removed. After the phosphorus is separated in a slag high in iron oxide that slag has to be removed in order that the bad effects of the oxides contained therein on the steel may be remedied. It is certainly difficult to see wherein lies the advantage of using costly electrical energy, electrodes, etc., to do the same thing that is done with cheaper fuel heating in a basic open-hearth furnace. And this is gradually becoming realized. After removing this first slag, it is replaced by a reducing slag to remove the oxygen from the metal. Here metallurgists have gone to the other extreme; they have aimed at and obtained the strongest reducing conditions possible by use of solid carbon and high temperatures. They have used as a measure of the intensity of such reducing conditions the formation of carbide in the slag. And these conditions are such as will readily reduce phosphorus compounds in the slag.

One method of avoiding two slags was proposed and tried out. It consisted in the use of a solid reducing agent, such as

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fine coke, added to the oxidizing slag after the phosphorus had been taken up by that slag. The object of this was to reduce the iron oxide of the slag and leave the phosphorus as calcium phosphide. The results of the tests on this process indicated that the use of a solid reducing agent at the elevated temperatures necessary in a steel-furnace not only reduces the iron, but also the phosphorus, and the latter goes back into the metal. No other result could have been expected. The explanation of the above reaction is simply that solid carbon has a stronger affinity for oxygen than has phosphorus at the temperature of molten steel in an electric furnace, even though the phosphorus be combined with lime as calcium phosphate. In other words, the reducing conditions acting on the slag are so strongly reducing that phosphorus goes back into the metal.

High-phosphorus iron-ore has been partly reduced in the presence of lime without practically any of the phosphorus entering the metal, the phosphorus being retained in a slag very high in iron oxide. Such reduction of iron and separation from phosphorus has been carried out in an electric furnace by using insufficient carbon to reduce the ore completely, since solid carbon, if present, will of course reduce the calcium and iron phosphates at the elevated temperature maintained. The action here of solid carbon is similar to that in a blast-furnace, in which the presence of an excess of solid carbon causes reduction of oxide of phosphorus and phosphorus is found in the metal. In an electric ore-reduction furnace, even with a basic lining, excess of coke will produce the same result, and the presence of lime does not overcome this effect. On the other hand, in an electric reduction-furnace, when the carbon-reducing agent is limited, the phosphorus does exactly what it does in an open-hearth—slags with iron oxide as iron phosphate, and combines with lime to form calcium phosphate after the carbon is used up by combination with oxygen of the ore. The above facts show why it is not possible to reduce iron from the first slag in the Heroult steel-refining furnace by use of coke, without reducing phosphorus back into the metal.

It is a fact that at an elevated temperature iron oxide is more easily reduced than calcium phosphate, and it is evident that if the proper intensity of reducing conditions be maintained at the elevated temperature, iron oxide may be com-

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pletely reduced without reduction of calcium phosphate. Such conditions are obtainable by use of a gaseous reducing agent, such as ordinary producer-gas, meanwhile maintaining the temperature electrically.

The various reactions of phosphorus associated with iron may be summarized as follows:

1. At temperatures below 1,450° C., phosphorus in pig-iron has greater affinity for oxygen than has the carbon in the pig-iron, but less affinity for oxygen than solid carbon in the presence of pig-iron.

2. At temperatures above 1,450° C., the affinity of the carbon dissolved in iron for oxygen becomes greater than the affinity of phosphorus in the iron, and the dissolved carbon can reduce calcium phosphate in the slag; of course, solid carbon can do this also.

3. Phosphorus oxidizes in presence of lime and iron oxide to calcium phosphate, in absence of silicon or solid carbon.

4. Silicon reduces calcium phosphate nearly always, but there may be a range of temperature below 1,450° where phosphorus oxidizes to calcium phosphate more easily than silicon to calcium silicate.

5. Solid carbon will reduce calcium phosphate contained in a slag or bath of iron, and phosphorus will go into the metal.

6. Calcium phosphate can form without oxidation of iron in presence of carbon dissolved in pig-iron at low temperature.

7. Calcium phosphate can form without oxidation of iron in absence of carbon and silicon at high temperatures; *i.e.*, above 1,450° C.

8. Iron oxide can be reduced without reduction of calcium phosphate contained in the same slag.

9. Solid carbon is a stronger reducing agent than carbon dissolved in the iron, probably because of the affinity of the metal for carbon.

It is apparent that in all of the present processes the complete control of at least one important factor is either lacking or has not been utilized. In the open-hearth process it is the control of the reducing conditions that is limited, and likewise in the Bessemer process; but in this latter process the control is further limited by the necessity of raising the temperature by oxidation of elements in the charge itself. In the electric

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furnace, however, where heat can be produced independently of combustion or chemical reaction, this important factor—namely, intensity of reducing conditions—can be controlled at will and with ease. Up to the present time this fact appears not to have been appreciated by metallurgists, judging by present methods. It is in the complete control of the reducing conditions that important possibilities of electric heating lie, and as this fact becomes more generally recognized and used it is probable that some method will be devised for measuring the intensity of reducing conditions or equilibrium conditions. Since the intensity of reducing conditions is the reverse of the intensity of the oxidizing conditions, the term "oxygen pressure" may find use as a measure of them.

Among the reactions made possible by the control of the reducing conditions and temperature simultaneously, is the reduction of iron oxide from a slag containing calcium phosphate without reduction of the phosphorus contained therein. Naturally, the origin of the phosphorus-containing slag or charge is immaterial; it may be raw ore containing phosphorus, or it may be a slag in which some of the original iron oxide served to provide the oxygen for combination with phosphorus, the oxide of phosphorus subsequently combining with lime to form phosphate. Or the reducing conditions may be so controlling as merely to prevent oxidation of iron, which, at the same time, causes phosphorus to oxidize and combine with lime.

In my experiments along this line it has been found that at temperatures below about 1,400° the phosphorus was easier to oxidize than carbon from pig-iron. And it was found that the phosphorus could be oxidized in the presence of lime without any resultant oxidation of iron and with practically no oxidation of carbon; that phosphorus could be oxidized in the presence of lime without oxidation of manganese; and a very interesting further observation was made, namely, that at certain temperatures between the melting-point of pig-iron and about 1,350° C., phosphorus oxidized in certain cases in the presence of lime without any appreciable oxidation of silicon. This would indicate a reversal of oxygen affinity of these elements—phosphorus and silicon in pig-iron in the presence of lime.

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A fusible slag results from proper proportions of acid and basic radicals. It has been found that the silica-content of the slag may be as high as 30 per cent. without apparently hindering the removal of phosphorus. Phosphorus was separated in slags containing this amount of silica and yet practically no iron oxide was present in that slag. This points to the possibility of a low melting-point slag for holding phosphorus.

The reduction of iron oxide from a charge or slag containing phosphorus as calcium phosphate, without reduction of the latter, requires essentially the same control of the conditions as does the selective oxidation of phosphorus in presence of lime, forming calcium phosphate without oxidation of iron. Electric heating is the only practical way of accomplishing such control.

It is not to be doubted that metallurgists will very soon appreciate the distinction between various degrees of oxidation and reduction, where they have not already done so, since in so many cases undesired elements can be separated from metals by oxidation of the impurity and reduction of the metal. When this becomes generally appreciated it is prophesied that electric heating will come into use to an extent that is now hardly thought possible.

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The Function of Slag in Electric Steel-Refining.*

BY RICHARD AMBERG, EAST ORANGE, N. J.

(Cleveland Meeting, October, 1912.)

WHILE the old-time melter was satisfied to bring his metal out in the desired condition, and therefore took care to have the slag liquid enough, the importance of a thorough knowledge of the slag is at present recognized in all up-to-date melting-shops. Formerly slag or cinder was considered a necessary evil, accumulating in some mysterious way wherever a man was striving to produce a clean metal. Nowadays we deliberately aim at a well-defined slag, which varies with the nature of the process and the product.

In steel-making we need slag for two main purposes, namely, refining and protecting the metal. To refine metal, essentially oxidizing and reducing processes are required, while to protect the metal we need a reducing or, still better, a neutral atmosphere. Although an absolutely neutral condition between steel, on one side, and the containing vessel and the slag, on the other, is an imaginary conception, the various furnaces in use for the production of liquid steel may be arranged with respect to the attainment of this ideal in about the following order of merit: converter, open-hearth, crucible, and electric furnace.

In dealing with the subject we will find it convenient to avail ourselves of the terms of physical chemistry, which yield a simple conception of the matter and an expedient way of expression.

Slag and metal at large are not mixable with each other, and we will therefore call them the two "immiscible phases," though there are certain compounds which dissolve in both phases under conditions to be considered later. Reactions be-

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tween chemical compounds can take place in either phase; they are independent of the reaction in the other phase as long as the product of the reaction remains in its own phase and does not cross the boundary-surface. A reaction of this kind, where the original and the final product are entirely in one and the same phase, is a homogeneous reaction. If the other phase takes part by either supplying or absorbing substances, the reaction is heterogeneous. The homogeneous reaction is governed by the laws of homogeneous chemical equilibrium or mass-action; it occurs uniformly throughout the entire phase.

Since, according to van't Hoff, at low temperatures an increase of 10° C. doubles the reaction-velocity, it will in most cases be immeasurably high at the temperatures of the steel bath. The heterogeneous reaction, however, must take place at the contact-surface of the two phases. Its speed depends greatly on the velocity with which the substances participating in the reaction are brought to the contact-surface and carried away from it. That is, it depends greatly on diffusion and mechanical convection.

Every condition which makes the slag less viscous and more agitable facilitates, therefore, a heterogeneous reaction. Furthermore, the greater the contact-surface between the two phases, the more is the heterogeneous reaction facilitated.

Some time ago[1] I pointed out briefly these facts, which are discussed more in detail in the present paper.

The above considerations will hold good for many kinds of metallurgical operations. When applying them to a practical case, we have to bear in mind that slag and metal are very seldom separated by a sharp surface; small particles of slag are hanging from the surface of the metal bath or cut off and scattered through it, and little balls of steel are often found within the slag. Considering a steel-furnace in operation, we have to deal with at least four phases, viz.: the slag and the metal, as the two liquid phases; the atmosphere above the bath, and one or more solids in the hearth of the furnace. The solid phases, however, can be disregarded entirely for the purpose of this paper, while the gas phase will occasionally have to be taken into account.

[1] *Metallurgical and Chemical Engineering*, vol. 7, No. 3, p. 115 (Mar., 1909), also *Stahl und Eisen*, vol. xxix., No. 5, p. 176 (Feb. 3, 1909).

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The function of the slag in an electric steel-furnace depends on the kind of work which the furnace is performing; that is, the particular process which is going on, and the stage which the operation has reached.

At places where cheap power is available, electric steel-melting begins with the most widely-known open-hearth process, that of melting pig and scrap. It is also possible to start with steel scrap without any pig-iron, although this is not advisable, since it is severe on the lining. On a basic-lined bottom and hearth, heated by one or more electric arcs, the charge melts down; slag is formed by lime, silica, and iron oxide in the form of ore or scale, and the refining by oxidation takes place through heterogeneous reactions of different kinds. After the ferric oxide dissolved in the slag has been reduced to ferrous oxide, it partly acts directly on those parts of the metallic solution which come in immediate contact with it, and partly dissolves in the metal, according to temperatures and coefficient of division of FeO between the two phases. If the slag is saturated with FeO, the latter can assume its full concentration in the metal bath according to the solubility-curve given in 1906 by Professor Eichhoff. Its action here on manganese and the metalloids is much the same as in the open-hearth, and needs no discussion at this time. It can be made more energetic, however, if the operator avails himself of a higher temperature, which is a special feature of the electric furnace, and, as Fig. 1 shows, of a higher concentration of FeO.

There is one essential difference from the reaction which takes place in an open-hearth slag. In a well-built electric furnace practically all the oxygen required has to be supplied in the form of solid oxide, while an unlimited supply of oxygen can be given to the open-hearth by opening its air-inlets wide enough to furnish an excess in its gases, from which the slag continuously and automatically derives what is required to replace the amounts transferred to the steel. I once operated a poorly-constructed electric furnace which produced such a tremendous draft of air through its doors that it scarcely needed more ore (to supply oxygen) than an open-hearth furnace.

During the period of oxidation the basic slags of an arc-furnace and those of an induction-furnace differ materially.

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Dephosphorization, a process which reaches its maximum velocity somewhat below the melting-point of soft iron, is accomplished more quickly in the comparatively colder slag of the induction-furnace than under the highest heat of the arc. At the present time, however, the technical significance of this fact is small, since the general—and, indeed, reasonable—tendency is to do as much of the refining as possible in the open-hearth furnace, and to reserve for the electric furnace merely those finishing-operations for which it is specially fitted, and which cannot be performed in the open-hearth.

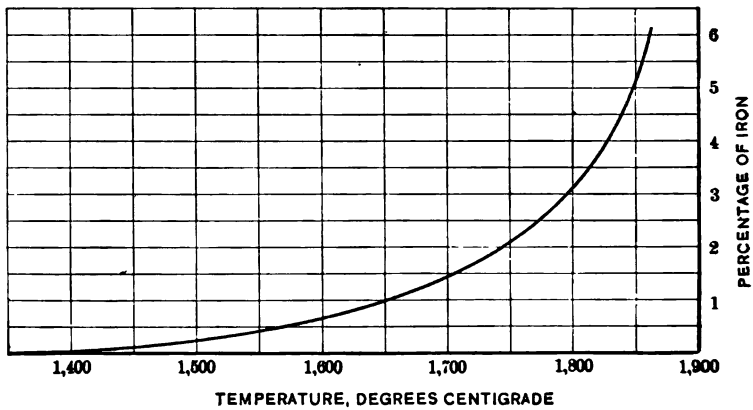

FIG. 1.—SOLUBILITY-CURVE OF FeO IN SLAG AT ELECTRIC-FURNACE TEMPERATURES.

In view of the above-described conditions, a comparison of the analysis of the slag during this period with that of open-hearth slag does not present any particulars of interest. CaO runs in the neighborhood of 40 per cent., sometimes from 5 to 8 per cent. lower; iron and manganese oxides, from 26 to 29 per cent. together; and silica varies with the silicon of the charge and additions. P_2O_5 may run up to any amount in crude open-hearth work, yet, as a rule, the slags have been skimmed when it reached something like 4 per cent. Manganese-ore, which even at this stage helps desulphurization, seems to act locally, perhaps near the electrodes only, in a manner which will be discussed later.

After oxidation has been carried on far enough, and the slag has been removed, or after dephosphorized metal has been

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transferred from another furnace to the electric furnace (which is the present usual practice in the centers of the steel industry), the metal will be deoxidized in the furnace. This involves a slag-process which none of the other furnaces can perform. Carbon is the chief deoxidizing agent, and is bound to act in both phases. Experience has shown that this element alone can complete the deoxidation, but it is not commercially practicable to let it do so, because it takes too long a time. The action of carbon on oxides in the homogeneous metal phase probably goes on with sufficient velocity, but its action on the oxides of the slag, being a heterogeneous reaction, is relatively slow. We know that glass can be colored by small quantities of carbon, yet we have not found that any amount of carbon worthy of consideration as a metallurgical reagent could exist as dissolved in the slag phase. Furthermore, when the metal has been freed from the compounds which are soluble in both phases, these compounds may tend to diffuse back from the slag into the metal as soon as favorable chemical and thermal conditions arise.

Phosphorus may be mentioned as an example. An iron phosphide exists in the metallic solution; by oxidation, transfer to the other phase, and neutralization with CaO, it forms calcium phosphate. In the finishing of a basic open-hearth heat the amount of oxygen in the slag decreases gradually and can reach a point where the other sources of oxygen are so far exhausted that the phosphate begins to be reduced by iron or manganese and the phosphorus re-enters the bath—one of the dangers which the skilled melter knows how to avoid. Here the electric method, as described in the patents of Humbert, offers the possibility of heating the slag, with the addition of carbon, in a reducing atmosphere, so quickly that it forms a considerably higher endothermic compound, probably Ca_3P_2, which is not reabsorbed by the steel phase; hence the opportunity to dephosphorize without skimming the slag. The sensitiveness of this reaction with respect to temperature is shown by the fact that, in the absence of very accurate control of the temperature, a rephosphorization of the metal has been found to take place.

Since in the reducing atmosphere, when this stage has been carried far enough, the "partial pressure" of oxygen is very

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low, the oxygen can be removed to a considerable extent from both phases without disturbing the equilibrium at the contact-surface, while the opposite would be the case in an open-hearth furnace. In fact, the amount of FeO present in the final slag is easily brought down in every-day practice to between 1 and 0.5 per cent., and, with a little attention, considerably lower. The reaction by which this is effected is a true heterogeneous one, and consequently does not come to an equilibrium in the original phase. FeO, as a base of the silicates or as dissolved in the slag, is freed from oxygen and the metal joins the other phase. Manganese is practically removed from the slag in a similar way. Even silicon is partly reduced when both the basicity of the slag and the temperature are high enough.

The desulphurization has been the subject of many experiments and discussions. No doubt there are three phases, the atmosphere and the two liquids, connected with this phenomenon. In the oxidizing period, where sulphur is being removed only to a small extent in the open-hearth, the electric furnace is much more efficient in this respect, especially when manganese-ore is used. It is likely that a considerable portion of the sulphur forms SO_2 and disappears with the gases; as the partial pressure of O and of SO_2 in the atmosphere of the electric furnace is smaller than in the open-hearth, the reaction takes place more easily. This is the reason why, as previously observed, the electric furnace in all other respects is inferior, as an instrument for oxidizing-refining, to the open-hearth.

In the reducing period a new desulphurizing action takes place, which I formulated some time ago,[2] as follows:

(1) $FeS + CaO + C = Fe + CaS + CO$ at high temperatures of the arc-furnace.

(2) $2\ CaO + 3\ FeS + CaC_2 = 3\ Fe + 3\ CaS + 2CO$ at still higher temperatures, where the calcium carbide can be formed.

(3) $2\ FeS + 2CaO + Si = 2Fe + 2\ CaS + SiO_2$ at the lower slag-temperature of the induction-furnace. A possible explanation is that FeS, distributed between slag and metal so as to produce equilibrium, takes part in one of these three reactions, new quantities diffusing from the metal into the slag, and so

[2] *Stahl und Eisen*, vol. xxix., No. 5, p. 176 (Feb. 3, 1909).

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forth. The reaction requires a high temperature, and even then takes much more time. This must be considered in connection with the possibility of reversing all the above reactions by introducing small amounts of iron or manganese oxides:

(4) $CaS + FeO \leftrightharpoons FeS + CaO$. This equation occurs from left to right for comparatively small amounts of oxygen, while from right to left it is the underlying principle of the three reactions (1) to (3).

In all the foregoing reactions one feature deserves the most important consideration, namely, the high basicity of the slag, which, in turn, is made possible by the high temperature attainable, especially in the arc-furnaces. The law of mass-action, which is directly applicable to homogeneous reactions only, describes the effect of the high concentration of CaO. Whatever silicates and aluminates of iron and manganese, of nickel, chromium, and other metals have been formed, they will be decomposed by its stronger base. This condition is also aimed at in the basic open-hearth. But it cannot be carried to the same extent as in the electric furnace, because the presence of more than 55 per cent. of CaO would result in a somewhat dry, viscous slag, and because the oxides thrown out of solution would not find a sufficient quantity of a proper reducing agent. From 12 to 13 per cent. of FeO and MnO is considered the lowest metal-content of an ordinary basic open-hearth slag for low carbon. Occasionally, and, besides, in the pig-and-ore processes, lower amounts may be observed.

The elements which, in the present electric furnace-practice, effect the reduction of these oxides, are chiefly carbon in the arc-furnace and silicon in the induction-furnace. As already observed, the manganese is practically eliminated; the FeO is easily removed down to 0.5 per cent.; nickel is reduced with great ease; chromium, tungsten, and vanadium, according to their inherent amount of free energy, with a larger consumption of power. Some of these elements are almost as liable to be oxidized by small quantities of oxygen as they are liable to be dissipated by volatilization, and in practical work some experience is required.

For explaining the mechanism of the reaction, we avail ourselves of the advantage which was referred to in the introduc-

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tory remarks, and which follows from our consideration of a system of various phases, so that we do not need to make any assumption as to what definite chemical compounds we have to deal with. It is usual, for instance, to consider open-hearth slags as solutions of the other constituents, such as CaO, $Ca_4P_2O_9$, Al_2O_3, FeS, CaS, etc., in iron and manganese silicates as solvents. In our case we are gradually decomposing the solvent, and are reducing its concentration to zero. It would not be of any advantage to know how the different constituents are combined with each other, even if it were possible at all to answer such a question for molten solutions.[3]

It is of great importance, however, for the advancement of general metallurgical knowledge, to determine the melting-points of the pure compounds, and give the complete diagram of the melting-points of their mutual solutions over as wide a range of concentrations as possible. Such work has been started in a splendid way by Vogt, Doelter, and others; but the data need to be much more abundant to make them suitable for practical use.

The melting-point (in degrees centigrade) of $FeSiO_3$ is 1,050; that of $MnSiO_3$, 1,150; that of $CaSiO_3$, from 1,220 to 1,125; and that of Mg_2SiO_4, 1,400.

Most of the open-hearth slags seem to keep in the neighborhood of the eutectic composition (*i. e.*, the one with the lowest melting-point) in those cases in which the compounds will form a eutectic. If one of the constitutents is raised extraordinarily to a high concentration, the melting-point as a rule goes up. Examples of this phenomenon are widely known. For instance, in the glass industry, the pure sodium- and the pure potassium-glasses as compared with the sodium-potassium glasses with a melting-point lower than each of the other two; mixtures of sodium and potassium carbonate, etc.

Acid slags become viscous when the silica is artificially increased; basic slags increase their melting-point with rising amounts of CaO, which can be easily raised more than 75 per cent.

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[3] Papers of James Hendrick (*Journal of the Society of Chemical Industry*, vol. xxviii., No. 14, pp. 775 to 778 (July 31, 1909); vol. xxx., No. 9, p. 520 f. (May 15, 1911), and others, though very valuable for agricultural research, prove that chemical analysis alone cannot solve the problems of the constitution of slags. Optical and thermal methods will have to be applied extensively.

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under the influence of the arc without any fluorspar in the slag. As this mineral, which is used to a larger extent in American than in European practice, is a good solvent, it prevents the use of an excessive temperature. It further permits us to keep the concentration in SiO_2 low. While from 12 to 20 per cent. of SiO_2 is often contained in the high-calcium slags, they can also be built up almost entirely from calcium, iron, and manganese compounds with very low SiO_2. One such slag, for instance, analyzed in its main constituents: CaO, 39.10; FeC, 18.20; MnO, 13.80; and SiO_2, only 2.10 per cent. Such slags may be gaining in importance for special work.

The carbon thrown into the furnace and the carbon derived from the electrodes can produce several compounds. Prevalent among these are silicon carbide, calcium silicide, and calcium carbide. This latter seems to be the most persistent under operating conditions, and is recognized by the development of acetylene from cooled-off samples. It is only after this carbide has been formed to a certain extent that the deoxidation of the charge can be relied upon as completed.

When this condition of the bath has been realized, the electric furnace represents the nearest approach to the ideal heterogeneous equilibrium between the different phases which has hitherto been accomplished in large-scale metallurgy. The converter and open-hearth are under the action of air and gas; in the crucible, the metal takes up carbon and silicon; while in the electric furnace the action of the metal on the basic lining is almost *nil*, and there is no exchange of elements between metal and slag. However, a small evaporation of the ingredients of the slag takes place, and in this respect the induction-furnace with its cold slag has a slight advantage over the arc-furnace. Under certain working conditions of operations of an arc-furnace, when it is thought advisable not to cover the electrode holes in the arch air-tight, fumes can be seen passing through the small joint between electrode and roof-brick. These fumes always leave a deposit on the cooler parts of the electrodes, in powder form. An analysis of them taken during the oxidizing period of a furnace showed, aside from minor quantities of other substances, SiO_2, 4.50; FeO, 8.49; Fe_2O_3, 60.60; CaO, 8.10 per cent. This composition

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allows the conclusion that after reducing the metallic oxides from the slag, there will be a considerable volatilization of CaO and some smaller volatilization of SiO_2, both being probably first volatilized as elements, and then oxidized in contact with the atmosphere. Smaller lots of Si may perhaps take part in the desulphurization by forming SiS_2, a volatile compound, and then decompose in contact with humid air or water.

In the foregoing I have discussed the points in which the chemical action of the slag of the electric differs from the corresponding slags of other furnaces. What is to be said of the physical effect holds good in general for both classes; but it is important enough to be considered in connection with my previous discussion. We have already seen that there is by no means an ideal two-dimensional surface between slag and metal, and that both phases penetrate each other to a considerable extent. This is due to the various operations which the charge has to undergo from the time of its melting. The boiling in the oxidizing period intermingles small particles and reverses their mutual position. Upsetting of the whole charge often takes place when pouring from one furnace into a transfer-ladle, and from there into another furnace. Then there are the effects of stirring, of making additions (which have to drop through the slag into the metal), etc. One might expect that a difference in specific gravities of from 3 to 7.8 would produce a clear-cut separation of the two phases by gravity; but the fact is that the separation proceeds with astonishing slowness, even at the higher temperatures and with the more liquid materials of the electric furnace bath. The prevailing belief that slag and metal behave like oil and water is only relatively true. The slag swims on the metal, but where small parts of the one are caught in larger masses of the other, they have to overcome an enormous friction in the attempt to separate. And the smaller a particle of slag is, the larger is its friction-surface in proportion to its mass, and the more time and effort will be required to overcome its adhesion to the glue-like iron walls which surround it on all sides. Following up this, there is every reason to believe that in dividing the particles of slag finer and finer, a point will be reached where the rising and separation of the slag particles would take such a length of time that all commercial advantage would be lost. If a stir-

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ring could be performed so as to agglomerate small particles into larger balls which would have a stronger tendency to rise, an important advantage would be gained. As long as we are not able to do this, we have to do the best we can by keeping the finished charge quiet, and at a sufficiently high temperature to allow it to settle. Hereafter, proper precautions have to be taken while teeming and pouring, that the painstaking results of refining shall not be reversed. But these matters are outside of the scope of the present paper.

This slow physical action may furnish a new explanation for the slow progress of the desulphurization as per equation No. 2 (p. 562). The microscope shows that siliceous and sulphidic products are sometimes contained down to two-thirds and even more of the distance from top to bottom of an ingot. These non-metallic products appear as balls in the ingot-structure, and are elongated into a cigar-like shape after rolling or forging the metal. Although their melting-point is lower than that of steel, they have not succeeded in uniting with the bulk of the slag in proper time. Consequently there is a lack of material in the slag, and desulphurizing can only take place whenever new sulphides are supplied. As the slag of the electric furnace allows us to hold the metal, the sulphur can be removed to any desired extent.

A few words, however, may be said on the conditions obtained when the electric furnace is lined with more or less siliceous materials. Probably as a result of the experiments of the late Thallner, the acid electric furnace has quite recently won some notoriety. An acid lining requires an acid slag to keep the hearth in good condition. This acid slag will be much the same as the slag of an open-hearth as long as the charge is treated with an excess of oxides and the heat is not too high. If necessary, it can be made to contain a higher percentage of SiO_2, for the reasons already set forth for highly-basic slags. The great difference from the basic electric furnace is that in the acid furnace all four phases are in lively reaction with each other, that conditions approaching a status of equilibrium cannot be reached, and that the metal must be "caught" at a certain moment for teeming.

The solid and the metal phase react in this way, that carbon of the bath reduces Si from the hearth, the amount of Si pres-

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ent being regulated by a reaction between the two liquid phases, namely, by keeping a sufficient stock of oxides in the slag to hold the Si within the required limits. The slag will therefore be a thoroughly black one, particularly during the beginning of the run. Thallner ascribes a specific beneficent effect to this exchange of Si and to the low heat-conductivity of the silica lining. Towards the finishing of the acid heat, the color of the slag clears up, and its reduction finally reaches a point where the glassy masses become light gray and green. Sulphur will, as in the acid open-hearth, remain unaltered in quantity, while phosphorus may be slightly decreased by phosphoric acid being thrown out of solution by the stronger SiO_2, and then reduced to phosphide of iron. This has to be confirmed, however, and it has to be explained how this reaction can take place. Nothing definite can be said about it at the present time.

It may be that some of the advantages of this method are due to the mechanical property of the slag, of agglomerating more easily than the basic slag particles to larger globules, which force their way up to combine with the bulk of the slag.

While dealing with a subject of such keen interest as this one, I need hardly say that a complete discussion of every detail is impossible and would conflict with my commercial obligations.

There are, however, so many points still open to discussion and to scientific research, that I would feel much gratified if this paper should act as a stimulus to further investigation.

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Heat-Losses in Furnaces.*

BY F. A. J. FITZGERALD, NIAGARA FALLS, N. Y.

(Cleveland Meeting, October, 1912.)

In any kind of furnace the question of preventing the loss of heat is important, for no matter how the heat is obtained it costs something; and consequently, other things equal, that furnace is most desirable which uses the greatest percentage of the total heat generated for the particular object in view. In the furnace which uses electricity for the generation of heat the importance of preventing losses of heat is very much greater than in any other. To generate 1,000 kg-cal. requires an expenditure of 1.16 kw-hr., and consequently the cost per calorie is always expensive when electricity is used for heating.

In heating any material in a furnace a general rule is that the generation of heat should be as rapid as possible. In any given furnace the heat-losses may be expressed as so many watts, and, therefore, the longer it takes to heat the furnace-charge the greater the losses will be. Suppose that a furnace contains a charge to be heated to 1,500°C. and that the average loss of heat is at the rate of 100 kw., then if the process of heating the charge takes 1 hr. the losses will amount to 100 kw-hr.; but if by using a more rapid generation of energy the charge can be heated in 30 min. the losses will amount to 50 kw-hr.

An example from experiments on a certain process may be given to illustrate this point. The problem was simply to heat a furnace-charge to a certain temperature, and experiments were made to determine the effect of increasing the rate of generation of energy and comparing the kilowatt-hours required to heat a given charge. Here are typical results:

Rate of Generation of Energy.	Energy Per Kg. of Product.
88 kw.	1.7 kw-hr.
93 kw.	1.6 kw-hr.

* Presented also at the Joint Meeting of the Institute and Section Xa, Electrochemistry, of the Eighth International Congress of Applied Chemistry, New York, September, 1912, and here published by mutual agreement.

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Thus by increasing the rate of generation of energy by 5.7 per cent. the energy-consumption was reduced 5.9 per cent.

This particular example is only a laboratory-scale experiment, but it serves to illustrate the point, that in the design of an electric furnace one of the first things to consider in avoiding heat-losses is to make the rate of generation per unit volume of charge as high as is compatible with other considerations, for in this way the heat-losses may be largely prevented. In the example given above the actual saving involved in working commercially, and expressed in money, amounts to $1,250 a year.

Another point of great importance in furnace-design in order to avoid heat-losses is to keep the external surface of the furnace as small as possible. This is so very obvious that it seems almost an impertinence to mention it, but any one who will take the trouble to study commercial furnaces of various sorts will find that this very point by no means receives the attention it deserves. Moreover, there does not appear to be any published information on what may actually be done in this way, so that the results of an actual experiment will be of interest.

In a resistance-furnace using 746 kw. the charge was heated to a temperature somewhat above 2,000° C., and maintained at that temperature for some hours. The furnace as originally constructed had an outside surface of 80 sq. m., excluding the foundation. The design was afterwards changed so that the outside surface was 53 sq. m. Determination of the energy-consumption per kilogram with the two designs of furnace gave the following results:

Surface of Furnace.	Energy Per Kg. of Product.
80 sq. m.	6.1 kw-hr.
53 sq. m.	5.0 kw-hr.
Reduction, 34 per cent.	18 per cent.

Having done all that is possible in the way of perfecting the design of the furnace so as to avoid unnecessary heat-losses, the final step is to provide the best heat-insulation possible. Heat-losses occur in various ways, but experience seems to show that the most serious losses are through the furnace body. Here the heat escapes in three ways: (1) conduction, (2) convection, (3) radiation. It is highly desirable that all these methods of loss should be studied, but at the present time the

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necessary data are lacking. A study of each source of heat-loss is difficult and will involve much time.

In view of these considerations it was thought desirable to make experiments as to the relative heat-losses which occur with various materials, some commonly used in furnace-construction and some specially made for heat-insulating purposes.

In order to do this furnaces were constructed which have been described in detail elsewhere. These consisted essentially of hollow cubes with sides of 230 mm., while the hollow spaces were cubes with sides of approximately 110 mm., so that the thickness of the walls of the hollow cubes was 60 mm. The furnaces were supported on knife-edges so that all sides were exposed to the air. Through small holes in the furnaces wires entered to carry electric current to resistors of nichrome wire, and other holes in the furnace permitted the insertion of a thermo-couple, by means of which the interior temperature could be determined.

In making an experiment a current was passed through the resistor and so regulated that the temperature was maintained constant. When equilibrium was reached the temperature and the rate of generation of energy were noted, the latter giving the amount of heat escaping through the furnace-walls at that temperature.

The results of several experiments are given in Table I.

TABLE I.—*Loss of Heat in Furnaces of Various Materials at Different Temperatures.*

Materials.	Temperatures Inside Furnace.			Remarks.
	500° C. Heat-Loss.	700° C. Heat-Loss.	800° C. Heat-Loss.	
	Watts.	Watts.	Watts.	
Fire-brick 1	460	760	910	Good quality.
Fire-brick 2	485	780	950	Good quality.
Fire-brick 3	425	705	860	Cheap brick.
Fire-brick 3 with insulating jacket 25 mm. thick	180	315	400	Special asbestos insulation.
Fire-brick 4	430	685	810	
Fire-brick 4 painted with lampblack	455	720	865	
Fire-brick 4 painted with aluminum	370	600	725	
Silica brick	565	920	1,100	Extrapolated at 800°.
Magnesia brick	860	.....	.....	At 400° C. loss was 580 watts.
Red building-brick	280	460	565	
Insulating building-brick	245	380	460	
Kieselguhr brick	130	220	275	

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The first column in Table I. gives the materials of which the furnaces were constructed and the second, third, and fourth columns give the heat-losses at 500°, 700°, and 800° C. respectively.

It is interesting to note in this table the effect of the nature of the outside surface of the furnace on the heat-losses in the case of fire-brick 4. It also appears that silica and magnesia brick are far from satisfactory insulators, and if they must be used in furnace-construction on account of their refractory qualities they should be backed up with a better heat-insulator.

Red building-brick and the special heat-insulating brick are fairly good insulators, but by no means as good as the kieselguhr brick. A comparatively thin jacket of specially prepared asbestos in the case of fire-brick 3 caused a very marked diminution in the heat-losses.

As regards the saving in heat-losses with large furnaces, one experiment may be quoted. The total outside surface of the furnace exposed to the air amounted to 20 sq. m.; the furnace was heated by means of a resistor, and it was possible to make determinations of the heat-losses at various temperatures. It was found that at 1,100°C. the loss amounted to 38 kw. Observation showed that there was considerable difference in the temperature of the outside walls. About 4 sq. m. of those parts of the outside of the furnace which were hottest were covered with the "insulating building-brick" referred to in Table I. and another determination of the heat-losses was made, which showed that these now amounted to 32 kw.

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Alloys of Cobalt with Chromium and Other Metals.*

BY ELWOOD HAYNES,† KOKOMO, IND.

(Cleveland Meeting, October, 1912).

As in organic nature certain animal and vegetable forms have undergone modifications, and thus, as it were, fitted themselves to live in a new environment, so it has been found possible in certain instances to form new metallic combinations which are practically immune to the natural conditions that exist on the earth at the present time. A few of the metals, such as gold, silver, and the metals of the platinum group, are found native, but the commoner heavy metals are nearly always found in the form of oxides or sulphides.

For a number of years I have made experiments with a view to finding certain metallic combinations or alloys which would not only resist oxidation and other harmful influences, but would also possess valuable physical properties, which would render them fit for special service. The first decisive step made in this direction was the discovery of an alloy of nickel and chromium in 1898. The properties of this alloy have already been described in a previous paper.[1] Immediately following the discovery of the nickel-chromium alloy, I produced an alloy of cobalt and chromium, which has likewise been described in the reference given, but the range of proportion between the cobalt and chromium is so great, and the combination is modified to such a marked degree by the introduction of other substances, that I have felt justified in reading another paper on this subject, describing more fully my researches in this particular direction.

As early as 1907 and 1908, I made alloys or combinations of cobalt, chromium, and tungsten; cobalt, chromium, and molyb-

* Presented also (by title) at the Joint Meeting of the Institute and Section IIIa, Metallurgy and Mining, of the Eighth International Congress of Applied Chemistry, New York, September, 1912, and here published by mutual agreement.

† Non-member. President of the Haynes Automobile Co.

[1] *Journal of Industrial and Engineering Chemistry*, vol. ii., No. 10, pp. 397 to 401 (Oct., 1910).

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denum; and cobalt, chromium, tungsten, and molydenum. I have made alloys of cobalt and chromium containing zirconium, tantalum, thorium, titanium, vanadium, etc. I have also added to the cobalt-chromium alloys the non-metallic elements, carbon, silicon, and boron. Some exceedingly interesting results have been obtained from these various combinations, and while further investigation is necessary in order fully to determine their chemical and physical properties, a number of them have shown interesting economic possibilities.

The preliminary fusions were made in graphite crucibles by means of a furnace using natural gas. I was later obliged to use crucibles of a special composition, not only to avoid the contamination of the metal by carbon from the crucible, but also because they proved more reliable under long-continued heating than those made of graphite.

The metal tungsten alloys readily with chromium and cobalt in all proportions. When added in small quantities to the cobalt-chromium alloy, it seems to have little influence on the properties of the combination, but if the proportion rises to 2 or 3 per cent. a notable effect is produced. Generally speaking, the cobalt-chromium alloy becomes harder and more elastic, especially if it contains a small amount of carbon, boron, or silicon.

The following experiment shows the effect of melting the alloy in a graphite crucible: 90 g. of cobalt, 6.3 g. of tungsten, 18 g. of chromium, together with a small quantity of calcium silicide, were introduced into a graphite crucible. The resulting alloy was very hard, and the crucible much eroded on the inside. The bar could be slightly flattened at one end, and after being made into a cold-chisel, showed remarkable qualities. It would not only scratch glass, but also quartz crystal. It was quite tough at ordinary temperatures, and would cut small chips or shavings from a piece of stellite. At a bright yellow heat it showed signs of fusion, and became covered with a skin of oxide.

An alloy was made by melting the following in a special crucible: cobalt rondelles, 80 g.; chromium, 20 g.; tungsten, 7 g.; calcium silicide, 10 g.; calcium carbide, 5 g. As soon as these materials were melted, the crucible cover was removed, and 15 g. of an alloy of cobalt and boron was introduced. The crucible

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lid was then replaced and the heating continued. A heavy, thick slag formed, which was removed before pouring the metal. The resulting bar was very hard and elastic, but only drew slightly under the hammer, and then broke. A cold-chisel made from the cast metal cut iron readily. The bar was broken up and remelted with about one-third of its weight of an alloy of cobalt, chromium, tungsten, and carbon. The result was a fine-grain alloy which was very elastic, and would draw out to a considerable degree under the hammer without checking. Its elastic limit must have been very high, since, when it broke, the pieces were thrown violently.

Taking the alloy of chromium and cobalt as a basis, and holding the proportion of chromium at 15 per cent. of the entire mixture, it was found that the alloy gradually increased in hardness with the percentage of tungsten. When the quantity of tungsten rises to 5 per cent., the alloy becomes distinctly harder, particularly when forged under the hammer. When the tungsten reaches 10 per cent., the metal still forges readily, and a tool formed from the alloy takes a fine cutting-edge. This alloy is suitable for both cold-chisels and wood-working tools. When the tungsten rises to 15 per cent., the metal can still be forged, but great care is necessary in order to avoid checking. This alloy is considerably harder than that containing 10 per cent. of tungsten, and is excellent for cold-chisels. When the tungsten rises to 20 per cent., the alloy is still harder, and can be forged to a small extent. It makes good lathe-tools for cutting steel and other metals at moderate speeds. When the tungsten rises to 25 per cent., a very hard alloy results, which cannot be forged to any extent, but casts readily into bars which may be ground to a suitable form for lathe-tools. These tools have shown great capabilities, particularly for the turning of steel, since they are very strong, and retain their hardness at speeds which almost instantly destroy the cutting-edge of a steel tool. The tungsten may be still further increased to 40 per cent., and the alloy will retain its cutting-qualities, and for turning cast-iron, this alloy answers even better than that containing 25 per cent. When the tungsten reaches 40 per cent. or more, the alloy becomes so hard that it will not only scratch glass, but will readily scratch quartz crystal. A small drill made of this material drilled a hole through the

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wall of a glass bottle without the use of any liquid or other lubricant. A $\frac{3}{8}$-in. square cast bar, when ground to a suitable edge, was set in a tool-holder attached to a lathe. The workman who had operated the lathe had been able to turn to form 26 cast-iron wheels in 10 hr. with a steel tool of the same size. The stellite tool turned 49 of these wheels to form in the same time. The steel tool was ground 50 times during the operation, while the edge of the stellite tool was dressed slightly by a carborundum whetstone, after its day's work was completed. A set of steel cutters, placed in the boring-head of a cylinder-boring machine, was able to bore from 27 to 28 holes in 10 hr. These cutters were replaced by others made of stellite, which performed the work in 3 hr. and 20 min., or a little more than one-third the time. Not only was the speed of the mill doubled, but the feed also, and notwithstanding this severe ordeal, the stellite cutters were only slightly worn, while it would have been necessary to regrind the steel cutters at least two or three times for the same service at slower speed. Some remarkable results were obtained in the turning of steel on the lathe. For example, a cylindrical bar of annealed nickel-chrome steel, about 2.5 in. in diameter, was placed in a lathe and turned with a steel tool at about as high a speed as the steel would permit without "burning." The steel tool was then replaced by one of stellite, and the speed at the same time increased to $2\frac{3}{8}$ its former speed. The stellite tool retained its edge under these severe conditions, and produced a shaving weighing $1\frac{2}{10}$ lb. in 30 sec. Just what the effect of the alloy will be in machine-shop practice, is at present somewhat difficult to determine. In my opinion, however, it will not fully supersede high-speed steel in the machine-shop, but in cases where rapid work is the main consideration, it will doubtless replace high-speed steel.

When molybdenum is added to a 15 per cent. cobalt-chromium alloy, the alloy rapidly hardens as the molybdenum-content increases, until the content of the latter metal reaches 40 per cent., when the alloy becomes exceedingly hard and brittle. It cuts keenly and deeply into glass, and scratches quartz crystal with ease. It takes a beautiful polish, which it retains under all conditions, and on account of its extreme hardness, its surface is not readily scratched. When 25 per cent. of

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molybdenum is added to a 15 per cent. chromium alloy, a fine-grained metal results, which scratches glass somewhat readily, and takes a strong, keen edge. Its color and luster are magnificent, and it will doubtless find a wide application for fine, hard cutlery. It cannot be forged, but casts readily, and its melting-point is not abnormally high.

If carbon, boron, or silicon be added to any of the above alloys, the alloy becomes much harder, though the addition is not always desirable, since it tends to make the alloys more brittle.

If either tungsten or molybdenum be added to a cobalt-chromium alloy containing 25 per cent. of the latter metal, the hardness of the alloy is rapidly increased. When the percentage of tungsten, for example, reaches 5 per cent., the alloy can be readily forged, and forms an excellent combination for wood-cutting tools, such as chisels, pocket-knives, etc. When molybdenum is added to the same mixture of chromium and cobalt, much the same effect is produced, though, generally speaking, a smaller quantity of molybdenum is required to produce a given increase in hardness. In some instances I have found it advisable to add both molybdenum and tungsten to the cobalt-chromium alloys. Generally, the color and luster of these alloys, after polishing, are brilliant, and they seem to resist atmospheric influences equally as well as the binary alloy of cobalt and chromium.

Discussion.

Philip N. Moore, St. Louis, Mo.:—Are these stellite tools cast and then ground, or are they forged and ground?

Mr. Haynes:—The lathe-tools are cast and ground. The knives are cast, forged at a temperature somewhat higher than that required for ordinary steel, and then ground. It is not necessary to temper or anneal them.

Carl Zappfe, Brainerd, Minn.:—Will these cobalt-alloys resist the action of hydrochloric acid?

Mr. Haynes:—Some of them are almost immune. Hydrochloric acid attacks them perhaps as readily as any other acid. I made one quantitative test, under the action of ordinary commercial hydrochloric acid, and at the rate of solution it would

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require about 15 years to dissolve a piece of stellite 0.75 in. long, $\frac{3}{8}$ in. wide, and $\frac{1}{32}$ in. thick. However, the alloy seems to be absoutely immune to nitric acid.

R. R. ABBOTT, Cleveland, Ohio:—There is a difference between the ability to cut a soft and the ability to cut a hard material. Is this stellite able to cut material harder than ordinary steel, or has it merely the property of being able to cut more rapidly?

MR. HAYNES:—Largely, I think, more rapidly; but it would stand up better than steel against hard material, as is mentioned in my paper.

S. T. WELLMAN, Cleveland, Ohio:—Were the steel tools used of high-speed steel?

MR. HAYNES: Yes; the best that could be obtained.

MR. WELLMAN:—Have you ever tried it on manganese-steel?

MR. HAYNES:—I have never tried it on manganese-steel. We did turn a bar of nickel-steel, however, that was rejected because it was said it could not be turned on account of being abnormal; but we put on the stellite tool and turned it very rapidly.

MR. ABBOTT: What is the tensile strength of stellite?

MR. HAYNES:—It varies greatly, the same as does the tensile strength of steel; but the test that I made about two years ago showed an elastic limit of 85,000 lb. and a tensile strength of 110,000 lb.

MR. ABBOTT:—How cheaply can this metal be produced as compared with high-speed steel?

MR. HAYNES:—It will cost very much more, on account of the initial cost of the material that is in it. However, we are getting the cost of cobalt down lower, and it will not add very materially to the cost of the tool, because it does a great deal more work than steel, and the tool is smaller; so the expense is not great.

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J. E. JOHNSON, JR., Ashland, Wis.:—Is cobalt more abundant than nickel?

MR. HAYNES:—I think not, except in Cobalt, Ontario, where there are very large deposits of cobalt-ores.

ALBERT R. LEDOUX, New York, N. Y. (communication to the Secretary *):—A prominent manufacturer of high-speed machinery, having seen the paper of Mr. Haynes which was presented at the Cleveland meeting, informs me that after a test he has adopted stellite lathe-tools throughout his plant. A comparative test gave the following results:

On phosphor-bronze, 125 ft. per min., using ——— superior high-speed steel; using stellite, 900 ft. per minute.

On tool-steel, 80 ft. per min., using ——— superior high-speed steel; using stellite, 133 ft. per minute.

On steel seamless tubing, using ——— superior high-speed steel, 100 ft. per min.; using stellite, 400 ft. per minute.

On cast-iron, using ——— superior high-speed steel, 100 ft. per min.; using stellite, 200 ft. per minute.

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* Received Dec. 2, 1912.

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The St. Helens Mining-District.

BY HORACE V. WINCHELL, MINNEAPOLIS, MINN.

(Cleveland Meeting, October, 1912.)

Location.

THE St. Helens mining-district, indicated in sketch-map, Fig. 1, is chiefly in Townships 9 and 10 North, Ranges 5 and 6 East of the Willamette meridian, in Skamania county, Wash. There is no railroad or even a wagon-road at present connecting it with any base of supplies; access is usually by wagon-road for

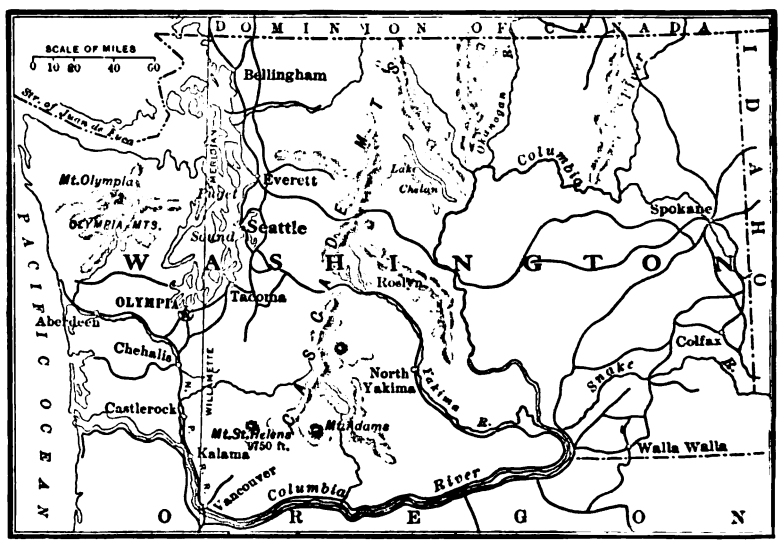

FIG. 1.—SKETCH-MAP OF THE STATE OF WASHINGTON.

about 50 miles from the town of Castle Rock to Spirit lake, and from that point by pony-trail from 6 to 10 miles farther.

History.

Nearly 20 years ago two German farmers, totally inexperienced in prospecting, penetrated this region upon a hunting- and fishing-trip. Their attention was drawn to numerous joints and crevices filled with black, glistening tourmaline, and con-

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taining interspersed crystals of pyrite, chalcopyrite, arsenopyrite, specular hematite, magnetite, and occasionally galena and zinc-blende. It seemed to them that they had stumbled upon a veritable Golconda, perhaps the center of mineralization of the entire region. After making as many mining-locations as possible, considering their limited supply of food, they returned to their homes with all the sensations of millionaires, and a determination to come again in the following spring and locate a few more bonanza mines.

From the enthusiasm born of ignorance, and solely through the reports and specimens of these two men, the prospecting-fever spread so rapidly that within three years there were 500 men in the district, and locations were made without number. The proprietors of a German newspaper published in Milwaukee became interested. They furnished the capital; and under the advice of their "expert," who was naturally one of the discoverers of the district, spent many thousands of dollars in prospecting-work. Mining companies of large capitalization were organized, and stock sold broadcast. Thousands of feet of tunnel were driven for the purpose of penetrating and exposing to view the enormous bodies of ores, the existence of which was supposed to be demonstrated by the tourmaline seams so abundant on the surface. The developments were not altogether satisfactory even to the dream-blinded journalists, and they finally concluded to take a moderate price by selling for a few paltry millions to an English syndicate. Then came the first inspection of the district by a man of some knowledge, and a report which suddenly stopped the expenditure of money. But the German prospectors were not to be so lightly divested of the rightful fruits of their discoveries. Others were interested, other companies were incorporated; abandoned claims were relocated and more stock sold, and in the course of time these mining-claims came to have a real value for the timber with which they were covered; and indeed some of it is wonderful. In order to secure this timber, mining-locations were plastered over the country in solid blocks with parallel end-lines and side-lines covering thousands of acres; and in order to keep up the pretensions of a mining-region, more tunnels were driven into the solid rock following the tourmaline-joints.

The total amount of development-work in the district

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amounts to about 11,000 ft., most of which I examined in the fall of 1911.

Topography and Climate.

The elevation ranges from less than 2,000 ft. above the sea-level to nearly 5,000 ft. The country in general is covered with a heavy growth of forest primeval. The undergrowth is abundant, and consists largely of huckleberry bushes, devil's club, and other shrubs, which frequently make it impossible to see any considerable distance ahead through the forest. The soil and wash covering the solid formation have a considerable depth. There is a mantle of pumice-stone, derived from near-by extinct volcanoes, which in many places has a thickness of several feet. The rocks are generally concealed from observation. The only places where exposures are continuous and clear for any considerable distances are the gorges of the mountain torrents and the tops of mountains, where the rocks project in steep bluffs. Sometimes such outcrops are found along and in the vicinity of Green river, which follows the main axis of depression throughout this district in a N–NW. direction. The snow-falls and rain-falls are evidently heavy, and the result upon the surface of the country-rock is that of a rapid erosion. There has also been in recent times a glacial mantle covering considerable areas, as is shown by the glacial striations upon surfaces of exposed country-rock. The mountain torrents usually flow over bare country-rock which is smooth and precipitous, and frequently marked by joints or slight steps. At other times it is almost as smooth as a toboggan-slide, making what is known in the Alps as a "glissade." These glissades are numerous and afford excellent exposures along a narrow belt of the underlying country-rock.

Geology.

Geologically, the country embraced within this district is comparatively recent. The main mass of the country-rock consists of diorite, a deep-seated eruptive of granitoid texture, consisting chiefly of plagioclase or soda-lime feldspar, and hornblende. With a very few exceptions, to be noted later, this rock is everywhere extremely fresh, comparatively uniform in texture, and seldom contains visible quartz crystals. Towards

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the northern portion of the district, however, quartz is noticed and the rock there is a quartz-diorite. In this diorite are many secondary or associated minerals. The so-called accessory minerals of marked importance and significance are pyrite, tourmaline, chalcopyrite, specular hematite, magnetite, and epidote. These minerals are in this instance not vein-minerals, but are accessory minerals belonging to the diorite itself.

The other rocks present include andesite and basalt. The andesite is later than the diorite and penetrates it in the shape of dikes. There are not so many or so largely developed accessory minerals in the andesite as in the diorite.

The basalt is found only in the NW. corner of the district. It is a dark-colored basic eruptive rock, fine grained, and perhaps the product of a more recent igneous activity which has broken out since the cooling of the diorite. No accessory minerals of any considerable quantity were noted in the basalt. Basalt and andesite tuff are neither of them formations of such origin, age, or history as to be particularly favorable for the deposition of ore.

Joints are fairly abundant, although over many portions of the district the interspaces are from 10 to 20 ft. broad.

Attractive to the Prospector.

Situated in a region of good climate, abundant water, and ample shelter, with structural material upon either hand, affording fuel at his door, the St. Helens district a few years ago, though generally unknown and mountainous, must have been decidedly stimulating to the imagination of the average prospector.

Upon first going to the St. Helens district he would observe in the water-courses and in the outcroppings of rock those indications of mineral which the surface affords, and nowhere any excavation to disclose what the underlying rock contains. The rocks present smooth, bare exposures, in which any structural features are plainly evident. There has been a slight surficial iron-staining in the vicinity of joints and crevices. There are upon the surface, scattered throughout the rock and associated with joints and other crevices, pyrite, chalcopyrite, specular hematite, magnetite, tourmaline, epidote, and occasionally arsenopyrite, galena, and sphalerite. All of these minerals in

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disseminated particles or in thin bands of segregation are of frequent occurrence.

Another favorable feature in connection with this district is the possibility of exploring to considerable depth by driving tunnels of no great length into the mountain-side at almost any point without the necessity of expensive machinery; and without having at his command financial resources of any considerable importance, the prospector is able by his own efforts to investigate the underground conditions and the significance and value of those indications which are found upon the surface. To the experienced prospector, the indications will not seem important or attractive, even upon its surface. To the inexperienced prospector, who is unable to distinguish specular hematite from galena, who cannot tell gray copper from a mixture of pyrite and arsenopyrite, who does not know the difference between chalcopyrite and tarnished massive pyrite, there would probably be an inducement to perform some work of exploration. He could even procure, as the result of hundreds of dollars' worth of work and weeks and months of labor, an occasional specimen for exhibit and even for assay, a specimen which might attract the attention of men with money, who themselves were unacquainted with mining and geology,—specimens which, lying in a broker's office, might sell stock to the uninformed. The prospector of experience would notice that there are no large placer-deposits. This is of peculiar significance and importance. In almost every region and mining-district where sulphide ore-deposits exist in large quantity, and where there has been superficial oxidation, decomposition of the rocks, and accumulation in the valleys of the products of erosion, the insoluble gold and silver of the ores have been accumulated in the form of metallic products in sufficient quantity to make placer-mining profitable. The St. Helens district is conspicuously devoid of such deposits. The country is a region of young topography. The valleys are new. They are not filled with any considerable quantity of gravel. Throughout the entire district which came under my observation, there is no indication of an accumulation of mineralized material or of free gold in placers.

Another thing which would strike the attention of the experienced prospector is the universal absence of quartz in vein

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outcroppings and in float. In other districts where large quartz veins occur, they are usually, although not always, harder than the country-rock in which they are inclosed. As a result, during the progress of erosion, the quartz veins project above the surface of the surrounding country and finally are worn down in fragments, and "float quartz" has been the means of guiding by far the larger number of prospectors to the veins which they are seeking and from which the quartz has been derived. No such quartz has been found in this district.

Still another fact would impress itself upon the experienced prospector, and that is the entire absence of gossan or iron-hat. Everywhere sulphide minerals occur in the rocks at the surface and in the joints within a few feet of the surface. So thorough has been glacial scouring, so rapid has been erosion, that oxidation has hardly begun.

The iron-hat, or gossan, frequently referred to as an important and valuable indication of ores in depth, is a zone of decayed and oxidized rock which covers the upper portion of veins and country-rock alike. The country-rock may not be rotted and oxidized to such a depth as the veins, but in any district where there is extensive gossan, the country-rock also is deeply rotted and oxidized. Now, this means that the sulphide minerals have been almost entirely removed. No iron-hat exists where fresh, bright, glittering pyrite and chalcopyrite minerals occur within a few inches of the surface. No gossan exists where the ferro-magnesian minerals of the eruptive rocks are perfectly fresh and sharp and bright, and where the feldspar shows no trace of decay. Furthermore, if the prospector were to enter this district to-day and see what is to be seen underground in many tunnels, he would discover no secondary enrichment. The natural, universal, and inevitable result of the oxidation of sulphide minerals, such as pyrite and chalcopyrite, in a vein or mass of country-rock, and its downward percolation into the rocks below, is the formation of secondary sulphide minerals. The principal secondary sulphide of copper is copper-glance, or chalcocite. This mineral is universally found where pyrite and chalcopyrite have been dissolved and carried downward through and along the unoxidized minerals below. The only possible exception to this statement just made is in a limestone-region

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where the carbonate of lime is so soluble and occurs in such quantity that it may precipitate from the sulphate solution all of the copper as carbonates. This seldom occurs, and even in our largest limestone copper-camp, at Bisbee, Ariz., although the carbonates are abundant immediately beneath the surface, the larger portion of the ore is secondary "glance."

Another thing which would impress the experienced prospector at the outset, and the average prospector if he saw the district to-day, is the universal fact that there is no improvement or increase of mineralization as depth is gained in the prospecting-work already performed. On the contrary, there is a general rapid diminution of alteration of the country-rock. There is no ore in commercial quantity. There are only specimens, and the specimens are obtained by handling and extracting many tons of rock for one ton of specimens. The last phenomenon which would impress itself upon the notice of the intelligent prospector is the absence of softening and alteration of the country-rock adjacent to the joints and seams. Wherever vein-mineralization has been extensive and important, and has effected the deposition of valuable ore-deposits, the adjacent rock is usually widely decayed and altered, not by oxidation, not by weathering, not by any surface-action, but through the solvent action of the solutions which deposited the ore in the veins. In Butte, for example, the granite and porphyritic rock inclosing the veins is altered, decayed, and mineralized with pyrite for hundreds of feet away from the main channels of mineralization. No such alteration occurs here. On the contrary, the country-rock is bright, fresh, clean, hard, and ringing within a few inches of the joints.

A prospector who has some knowledge of mineralogy and of the origin of minerals would be impressed by one or two additional facts. The joints are associated with and enlarged by deposits of epidote, tourmaline, and magnetite. Tourmaline is a mineral formed from vapors. It is what is known as a fumarolic or pneumatolytic mineral. It is not formed by ordinary vein-waters. It occurs in the vicinity of and along joints, and it was deposited before the rocks were cold, and possibly before they were sufficiently cool to enable vein-solutions to exist. The chalcopyrite in this entire district is so universally associated with this mineral and with specular hematite, magnetite,

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epidote, and the usual concomitant sulphide minerals, such as arsenopyrite, sphalerite, and galena, that there can be no question whatever as to the date and manner of deposition; the origin of these sulphide minerals in the joints and associated with the joints is pneumatolytic. Pneumatolytic copper-tourmaline ores are rarely deposited in sufficient quantity to make important copper-mines. They represent the initial stages of mineralization, and where they still occur in their original character and position there is seldom anything of value.

There has been one other form of deposit of copper and iron sulphides in the St. Helens district: that is, purely magmatic segregations of sulphide from a molten mass, just as matte separates from the slag in the slag-pot. This form of deposit is exemplified at the Polar Star, where there are the usual blotchy and scattering bunches of sulphide minerals in the solid unaltered country-rock.

The prospector of judgment and experience must be a man of some knowledge of geology and mineralogy. He must have some conception as to the conditions under which ores can be worked profitably, and how much ore it takes to make a mine. Such a prospector going into this district to-day could not fail to see on every hand evidences of disappointing results and of general barrenness.

In the preparation of the above description, from 25 to 30 different claims and groups of claims were examined. The amount of development varied from several hundred feet of tunnel and drifts to small open pits. The maximum showing on which this work was based was a brecciated zone 6 ft. wide; the minimum was a mere joint-crack. In the commonest experience there had been revealed from 1 to 2 ft. of brecciated or sheeted rock, with small but practically barren veinlets. At the best, after hand-sorting many tons of the run of the mine, comparatively few pounds of low-grade copper sulphides were obtained and other ores are almost entirely absent.

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The Occurrence of Gold in the Eocene Deposits of Texas.

BY E. T. DUMBLE, HOUSTON, TEX.

(Cleveland Meeting, October, 1912.)

FOR many years there have been occasional reports of the discovery of gold from a belt of the coast country of Texas which is underlain by deposits belonging to the lower Eocene. For the most part they have been treated as newspaper reports and no examination or record of them has been made. Among such, it may be mentioned that in 1896 there was a flurry over reported gold-discoveries west of Luling and elsewhere in this belt, and in 1903 some prospect-shafts were sunk on the Gonzales branch of the Galveston, Houston & San Antonio railway, 3 miles below Harwood. During 1912, further discoveries have again brought the matter to our attention, and it may be of interest to record the facts as we have found them.

Some years ago specimens were sent me, which were said to have been taken from certain beds on Gazley creek, and concentrated by washing in a rocker. In them were several particles of gold sufficiently large to be visible to the eye. Under the microscope they showed to be, with few exceptions, sharp cornered, and not rounded as is usual with placer-gold. They were not filings, however, but their sharpness seemed rather to be due to a recent freeing from their matrix. One or two grains were rusty, and one copper-hued, but the most of them were tolerably bright.

On examining the material of the samples small grains or fragments were found, of a yellowish brown color, which seemed to be of a finely-granular structure—possibly a sandstone. In these fragments I found particles of gold, one of them being large enough to be visible without a lens.

Gazley creek empties into the Colorado river just north of Smithville, and for a few miles north of that place its banks show, beneath the deposits of more recent materials, the sands and clays of the Lignitic division of the Eocene, with lignite, shells of iron, etc.

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A characteristic section is as follows:

Bed No.		Feet.
1.	Detritus covered,	15
2.	False-bedded sands with iron, the seams and lamina being indurated ferruginous sandstone,	4
3.	Similar to No. 2, but less hardened and containing some seams of chocolate clay,	3 to 4
4.	Lignitic sands and clays with some lignite,	1 to 3
5.	Gray clay with shells of iron, to water,	1 to 3

It was from these beds that the specimens sent me were supposed to have come.

I spent two days along the creek securing representative and average samples from each of the beds and in panning dirt from various parts of the beds and from the creek. In panning, only a very few colors were seen, and these were from beds Nos. 3, 4, and 5. I also found in the beds fragments of sandstone similar to those in which the gold occurred in the original samples, although I did not find gold in any of my sandstone fragments.

Very careful assays were made of the samples I collected, with the following results:

Sample No.	Location.	Gold.
1.	Upper part bed No. 3,	None.
2.	Middle part bed No. 3,	Trace.
3.	Lower part bed No. 3,	Good trace.
4.	Bed No. 4,	Good trace.
5.	Bed No. 5,	Faint trace.
6.	Iron-seam, bed No. 5,	None.
7-*a*.	Iron-seam, bed of creek,	Strong trace.
7-*b*.	Iron-seam, bed of creek,	None.

The only visible gold in the samples taken by myself was found in No. 4, in which several small particles were clearly seen under the lens.

The actual occurrence of gold in these deposits was therefore proved, although I was unable to find it in such quantity as was indicated by the original samples.

The work south of Harwood was brought about by finding in wells in that vicinity a stratum of fine argillaceous sand mixed with glauconitic matter and carrying quantities of small crystals of pyrite which showed gold when assayed.

The sand-bed, reached at 200 ft., was 80 ft. in thickness.

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Assays of carefully-taken samples made by reputable assayers showed values of $20 and upward. Two shafts were started and were sunk to more than 100 ft. in depth, but, as I understand it, owing to trouble with quicksand below that depth they were abandoned before the supposed gold-bearing sand was reached.

Later, parties digging a well southwest of Harwood were attracted by the appearance of the materials found, and sent samples to the U. S. Assayer at El Paso, who reported very favorably on them. The well was then enlarged and deepened, and carefully sampled. The assays, made of materials taken every foot from the bottom upward for 38 ft., all showed the presence of gold, although only as traces in six specimens. In the others the gold-values ranged from 16 cents to $8 per ton, with small amounts of silver additional.

Samples of these sands from other wells in this vicinity also show small values in gold.

In both these places the geological relations are the same as those in the Gazley Creek region, so that all occurrences which have come under our notice are connected with beds of the Lignitic stage (Chickasawan) of the Eocene.

The materials of the Lignitic beds in this area are interbedded clays and sands with beds of lignite. The sands are quartzitic, more or less micaceous and carry occasional small pieces of feldspar. They also carry a considerable quantity of iron in the shape of scattered grains and nodules of pyrites and some ferric oxide.

The composition of these beds immediately suggests the granite area of the Llano region as the place of origin of the materials entering into them, and the channel of the Colorado river or its early Tertiary predecessor as the route by which they could have reached the shores of the Lignitic lagoon or swamp in which they were deposited.

Gold and silver are known to occur in the porphyries and micaceous schists of the Llano region, and gold is found among the sands lying in the creeks flowing over or through these rocks. Considerable quantities of placer-gold have been found in the Big Sandy, and it is found in the sands of the Colorado as far south as Austin.

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The later Tertiary deposits of the Colorado basin south of Lagrange show quantities of similar materials derived from the Llano region, the fragments of feldspar and granite being of much larger size than any that enter into the Lignitic beds. Consequently, while grains of free gold might not have been transported such distances as are here involved, fragments of the sandstones, porphyries, or mica-schists with gold inclusions might have been so transported and by subsequent disintegration have freed the gold.

It would, therefore, seem entirely probable that the Llano region, with its granites and porphyries, and its gold-, silver- and iron-ores, was largely uncovered in early Eocene times and was subjected to active erosion, the detritus from it being swept coastward by waters running into the Colorado and deposited as the beds we now call the Lignitic.

These deposits are now known to carry gold locally over a region about 60 miles in length, and while it may well be that the colors do not exist at any place in so concentrated a condition as to make it possible to work them, the fact that the gold does actually occur will doubtless induce repeated attempts to do so. Furthermore, the nature of the occurrence is of sufficient interest to warrant a much more extended study than has yet been given it.

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Present Conditions of Mining in the District of Vladivostok, Siberia.

BY ALBERT F. J. BORDEAUX, THONON-LES-BAINS, FRANCE.

(Cleveland Meeting, October, 1912.)

THE immediate vicinity of the sea-shore, affording special facility for the exportation of ores, makes it possible to work certain mines in the Vladivostok district, which, in more remote places of Siberia, would remain unproductive. Outside of the city of Vladivostok the population is small and the country is very little prospected. Additional discoveries of valuable deposits like the large body of calamine which has recently been opened in the neighborhood of earlier worked lead-silver ores, in the Tetiuhe valley, are still in the field of possibilities.

For many years gold alone was produced, in part from the coastal rivers and in part from a quartz-vein on Askold island near Vladivostok. But the production is diminishing, and prospectors are looking for other rivers in the interior, such as the tributaries of the Ussuri. These rivers are now more accessible than formerly, due to the slow but gradual development of colonization by the Russian peasants. The few natives of the country, called Orodjones, are of the Chinese race. They resemble the North American Indian and are hardy fur-hunters. They have given some attention to placers and have already shown good specimens of gold-dust from the rivers Iman, Bikin, Noto, etc.

Coal-mining was begun in the Vladivostok region more than 20 years ago, but in a somewhat irregular way. The industry is not as yet important and the best coal-seams are badly prospected, or are even neglected.

Copper, lead, and iron exist in several places. The amounts give some encouragement to the anticipation that the discoveries will give rise to important metallurgical works.

I here present a review of the various deposits, based upon such safe information as is in hand, when I have not drawn upon my personal observations. Fig. 1 is a map of Eastern Siberia, showing the mineral deposits.

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Gold.

During the past two years, the quantity and value of the gold-production of the Ussuri, or Vladivostok, district has been as follows. The area included extends north only as far as the Tumnine river, 600 miles from Vladivostok.

	Quantity.		Value.	
	1909. Kilograms.	1910. Kilograms.	1909. Per Ton.	1910. Per Ton.
Tumnine river, ...	44.611	11.774	$1.20	$2.40
Nikolaevsk-Tumnine, . .	80.438	20.260	1.50	1.95
Askold, sands, ...	0.433	2.524	0.40	0.18
Askold, quartz, ...	15.994	6.185	3.95	1.25
Ussuri bay, ...	0.911		0.55	
Total, ...	142.387	40.743	for 77,652 tons.	

The production is small, owing to the present exhaustion of the coastal rivers of easy access. The mining, moreover, is done in the most primitive way. For the quartz of Askold, there is a 5-stamp battery. The total production of this mine for 30 years is about 850 kg., valued at $500,000.

Lately, prospecting has begun on the Iman river, and as a result, 7.5 kg. of gold has been obtained from 2,044 tons of sand. Gold exists in the small creeks, and also in the Iman itself, and on the terraces of a probable glacial formation, that borders the river. Dredging, as well as hydraulicking, may in time be employed, since the surrounding hills, more than 300 m. high, give sufficient fall for hydraulicking purposes.

The natives have found gold in the Bikine, Noto, and Khor rivers. There is gold also along the Manchurian boundary.

There are some quartz veins on the coast, opposite Askold, and also at Sedimi and Adimi, on the other side of the Amoursky zaliv (gulf), 40 miles from Vladivostok. The veins contain arsenical pyrite, and antimonial copper sulphides, in a gangue of schists. Large outcrops of hornblende-granite, and a black igneous rock, probably a gabbro, occur close by.

Calamine, or Zinc-Ores.

The discovery of calamine in Far-Eastern Siberia was made by me, near an old Chinese lead-silver mine. The old diggings were about 10 m. deep and very irregular. Probably the Chinese had a way of their own of separating lead from silver by

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oxidizing the lead sulphides. During the smelting-process the greater part of the lead seems to have been lost.

Prospecting had been done by Bryner & Co., of Vladivostok, since 1897. This firm, in 1902, showed me a few specimens of

FIG. 1.—MAP OF EASTERN SIBERIA, SHOWING MINERAL DEPOSITS.

ore, from which I inferred the existence of calamine. Not until 1903 was I able to inspect the place, and then the discovery was an easy matter. Only one familiar with mineralogy would have suspected the value of this white rock.

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The Tetiuhe deposit is about 250 miles from Vladivostok by sea, and 21 miles from the sea-shore. The distances are such that with sufficient tonnage mining may be profitably carried on. After two or three months of prospecting, more than 10,000 tons of ore with 48 per cent. of zinc was demonstrated.

Unfortunately, the Japanese war, which began in 1904, made the resumption of work impossible until 1907. A road was then built and about 900 tons of calamine was shipped to Europe. During 1908 and 1909, a 2-ft. gauge railway, about 36 km. long, was constructed, by the firm of Arthur Koppel. The loading-facilities in Titiuhe bay were improved by a pier. At the mines, according to Russian custom, schools, hospital, bath-house, library, and many other buildings were erected. A double-track gravity cable-tram joined the mine and the railway. The telephone and telegraph connected all departments and the manager with Olga bay, and thus with Vladivostok. There are more than 1,000 workingmen, of whom two-thirds are Chinese and Coreans. At Titiuhe is the largest mining-camp in the Vladivostok district.

By the end of 1911, more than 55,000 tons of calamine, averaging 43 per cent. of zinc, had been shipped. There is still more than 50,000 tons of ore in sight, probably containing more than 40 per cent. of zinc.

Like similar calamine-deposits, the Tetiuhe ore appears in lenses and masses of irregular shape in a metamorphic limestone. Such lenses are called *griffons* in Greece, because they extend like fingers into the inclosing rock. It is a metasomatic, or replacement, deposit. The zinc carbonate has replaced the calcium carbonate by the reaction of acid solutions containing the zinc sulphide, or blende, upon the limestone. The zinc-blende and galena adjoining the calamine constitute a contact-deposit along the border of a huge porphyry sheet with underlying limestone. In the porphyry itself, there are some lenses of the sulphides, galena, zinc-blende, and pyrite. Apparently some silicic acid derived from the porphyry has given rise to some zinc silicate, which is mixed with the calamine. As copper is present, the calamine is sometimes tinted with the blue carbonate and the red oxide of copper, so that it resembles the Mexican *herrerite*.

Among the metasomatic minerals produced along the con-

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tact of the porphyry, there is a large quantity of tremolite, crystallized in the shape of needles radiating from a center. It is very hard, and has various colors, from red to green, according to the proportion of iron present. After long exposure, the tremolite undergoes decomposition and becomes very soft, and being intermixed with galena and zinc-blende, it is a serious drawback in the separation of the sulphides. Among the contact rocks, on the top of the deposit, there are also some quartzites and some black metamorphic slates.

The porphyry is a quartz-porphyry with biotite as the dark silicate, and with some crystals of zinc-blende. It has a dark-green color, is very compact, and often looks like andesite or one of the more recent forms of dioritic rocks. On the upper parts of the hill, there is a gray porphyry which seems to be an alteration-product of the green variety.

The limestone is Jurassic in age. Fossils are scarce, yet a *Cidaris* and some mollusks have been found. The latter have not been accurately determined. The Jurassic beds run parallel with the Sikh-ot-alin range, but with many interruptions. It might be of interest to prospect this range of mountains to the north towards Saghalien island, and to the south along the intersection with the sea-shore. In the latter area the calamine-deposits, if there are any, would be more advantageously situated. But so far, outside Tetiuhe, zinc carbonate has not been discovered.

There is a small deposit of zinc silicate near Olga, 50 miles south from Tetiuhe. This is also a contact-deposit in connection with iron-ore instead of lead-ore, and it will be more properly described with the iron-deposits. The quantity does not seem to exceed 2,000 or 3,000 tons of zinc-ore.

From the above description, it may be inferred that there is a possibility of more discoveries of calamine along the Coast range, as was the case with similar deposits in Sardinia, Tunis, Mexico, and elsewhere. If found, their development and mining would be much helped by the experience gained on the Tetiuhe mines. It may be said here that the freight to Europe from the Tetiuhe bay is only 25 shillings, or $5, per ton. It may also be remarked as an important consideration, that the price of calamine in Europe (Antwerp, Hamburg, or Dunkirk) fluctuates with the market: a rough ore with 43 per cent. of

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zinc, and with the zinc at £25 a ton, sells at £5.10, or about $27.50, per ton.

Although the lead and zinc sulphides of Tetiuhe are part of the same deposit, they will be described separately, as being entirely different ores and requiring other methods of mining and treatment.

Silver-Lead Ores.

The main silver-lead deposit at present is the Tetiuhe mine, Fig. 2. The silver-ore was the occasion of the discovery of the calamine. Prospecting-work dates back to 1897, but the mine

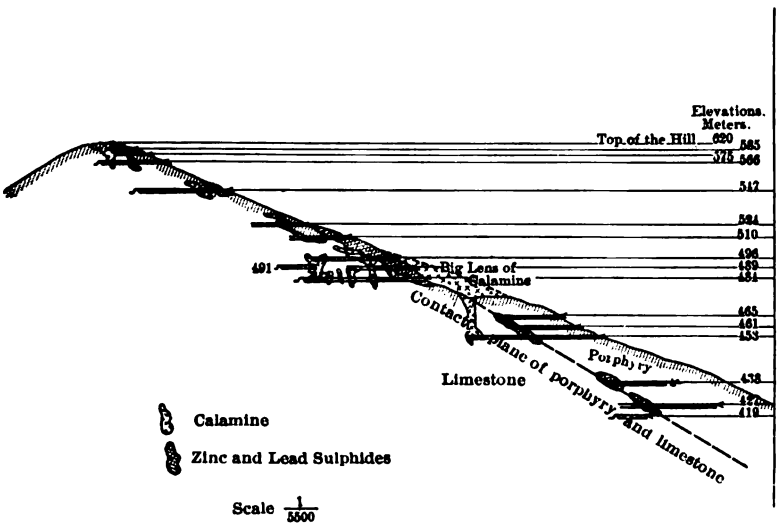

FIG. 2.—SKETCH OF THE MINE AND WORKS OF TETIUHE (SIBERIA) ZINC- AND LEAD-DEPOSIT.

was developed only after 1907. The ore is an intricate aggregate of the sulphides—galena, zinc-blende, pyrite and gray copper, in a gangue of tremolite (actinolite). The different constituents are so closely mixed that no hand-sorting is possible. The proportion, moreover, of silver does not vary so much with the lead sulphides as with the copper sulphides (or gray copper), and as this last constituent is scarcely visible, being so scarce, no separation can be made except after fine crushing.

I have previously described this deposit as a contact between the limestone and a blanket of porphyry which appears towards the outside of the hill, making possible the opening of the ore

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by short adits. The contact-lenses are most developed, but in the lower part of the mine there are, besides, a few segregations of sulphides in the igneous magma itself (magmatic inclusions). The constituents are the same, so that one may infer their close relations to the contact-lenses. A deposit of this kind is of course very irregular, but the type is not scarce in Siberia, where many lenses of lead-silver ores have been worked in the Altai and Nertchinsk districts, as will be described hereafter.

The total amount of underground works at Tetiuhe for developing the calamine, and simultaneously the lead-ores, is about 4 km. of tunnels in 16 levels on a total elevation of 170 m. These figures include neither the shafts and stopes, nor the big open-cuts for the calamine. All the great stopes are practically development-work for the lead-silver ore-bodies.

The total amount of sulphides thus developed may be estimated roughly at several hundred thousand tons, and the question at present is how to make the most of them. The first experimental runs for a concentrating-plant have been made in Europe, with the ultimate plan of erecting a plant at the mouth of the Tetiuhe river. The treatment of various similar deposits in the neighboring valleys of Tatushe, Olga, Preobrajenie, Djighit, etc., would eventually give rise to a custom concentrating-plant, as in many districts of the United States and Mexico.

There are other lead-silver deposits of the same kind in Siberia. The one most closely similar, and also the nearest, is that of Nertchinsk, where some calamine, or zinc silicate, was discovered long ago. The total number of mines at Nertchinsk has reached 448, of which 306 are in the limestone. There were 14 main deposits. The largest have produced 170,000 tons. The average proportion of lead has been 2.69 per cent., and the silver 283 g. (9⅓ oz.) per ton, in a total production of 1,457,000 tons, extending over 167 years, from 1703 to 1870. Recently the production has fallen to an almost insignificant amount.

It is not at all impossible that the district of Vladivostok may also become an important producer of lead, with the eventual erection of a smelting-plant at some convenient place.

It should be noted that the smelting-furnaces of small ca-

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pacity, which could be erected in former times, are now no longer possible. The cost of labor is much higher, while at the same time the value of lead and silver has greatly diminished. Plants of large capacity, although expensive to build, can be worked economically when the general expenses are distributed over a correspondingly large tonnage of ore.

Copper-Deposits.

Copper was known to exist in the Vladivostok district as early as 1902, at the time of my first visit, but even now, in 1912, the amount of prospecting is small. Yet the copper-deposits are really of interest. In some places they are true fissure-veins in andesite. In the Djighit valley, north of Tetiuhe, they are not far from the sea. The thickness of the veins is about 2 ft., and the vein-matter is quartz and copper pyrite. The location, about 100 m. above the river, is very favorable, and the proportion of copper seems high.

There are also some copper-ores (gray copper) in the limestone of the Tetiuhe valley, along the contact with the porphyry. Some copper-ore also appears in the lead-zinc ores of Tetiuhe. Such deposits, while less interesting than veins, may yet possess importance should the copper-smelting industry take root in the district.

There is a premium of £12, or $60, per ton of copper produced in Russian territory, and there is also a premium of £4. 5*s*., or $21, per ton of lead. This award was evidently made to encourage the smelting industry in Eastern Siberia.

There is some iron-ore near Nicolaevsk, at the mouth of the Amur river, but the locality is outside of our district. In the Vladivostok district, there are two iron-deposits, a short distance from each other, the one near Olga bay, and the other near Vladimir bay. They are segregation- and contact-deposits, in and between granite or porphyry and limestone. The latter has been metamorphosed to marble, with a thin crust of calamine, or zinc silicate, which must have been derived from small patches of zinc sulphide in the vicinity.

The ore is magnetite, either pure or mixed with garnet and actinolite or tremolite, the same contact-minerals which appear at Tetiuhe. Titanium also was found in a few analyses. That

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there are some sulphides in the neighborhood, I infer from the Chinese shafts, which have not yet been reopened, but which were very likely sunk for silver-lead ores. This magmatic ore seems to have originated from the porphyry, which is more ancient than the granite. The granite breaks through the porphyry dike.

An estimation of tonnage is difficult, owing to the irregular shape of the deposit and the lack of sufficient development. Some have thought that both deposits together may contain 1,500,000 tons.

The Vladimir deposit is a little the more developed of the two. There is a tunnel 100 m. long, and there are several shafts. The ore is magnetite, and ranges from 45 to 53 per cent. of iron. Rocks and minerals like those at Olga outcrop around the deposit.

Chromium, Platinum, Quicksilver, Manganese.

There is a chromium-deposit in the northern part of this Ussuri district, which, conformably with the usual experience, is in serpentine, but no development-work has been done upon it.

The manganese-deposits which appear in connection with limestone, have also not been developed.

Platinum was discovered at the mouth of the Anadyr river many years ago, but probably the alluvial sands are unworkable. More recently, iridium has been found in one of the coast rivers, with the alluvial gold. It lies close to the Manchurian boundary, where I personally examined the occurrence. Quicksilver has been reported in the Anadyr valley.

Coal-Deposits.

There are numerous coal-seams in the Vladivostok district, including Saghalien island. The coal-measures seem to exist all along the Sikh-ot-alin range, which runs parallel to the coast until cut out in its southern end by the sea, making the salient *rias* of the Vladivostok coast. The Sikh-ot-alin is a folded chain. The strata are dislocated, especially the oldest formation, a fact liable to affect also the coal-deposits. There are three distinct coal-formations: the lignites, the Mesozoic and Tertiary coals, and the anthracites.

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The lignites are worked about 25 miles from Vladivostok; the production was 107,000 tons in 1910, and about as much annually in several preceding years. The Mesozoic and Tertiary coals are found on the island of Saghalien. There, before the Japanese war, in various places the production was from 25,000 to 30,000 tons a year; the output diminished to 11,500 tons in 1910. The seams are much tilted, in fact almost to the vertical. The most ancient horizon is the upper Cretaceous (Senonian), but the largest seams are in the lower Tertiary, adjoining the Cretaceous. They follow a N–S. line, about 90 miles long. The Miocene and Pliocene formations also contain some coal-seams. Among the best of the Saghalien coals is the product of the Pilevo mine, containing from 70 to 85 per cent. of carbon. Pilevo enjoys, moreover, the rare advantage of a partly-protected harbor, while the whole western coast of Saghalien, with its coal-mines, has no harbor.

The anthracites occur east and west of Vladivostok. The most important mine at present is Su-Tchang, which belongs to the Crown. This mine, about 85 miles from Vladivostok, is connected with the city by railroad. The coal is somewhat poor in quality, and is very brittle. Its calorific value is 25 per cent. lower than that of Cardiff coal. This coal is used by steamers, because there is nothing else to be had, but it is far from satisfactory. The annual production of Su-Tchang is between 120,000 and 200,000 tons. The working-costs are very high, and the mine would probably be abandoned if another deposit of superior quality were available.

New developments are probable at Mangugai, where there is a coal containing 60 to 75 per cent. of carbon. Mangugai is 24 miles from Vladivostok, half way by sea, and half way by land. Tests have been made in comparison with Japan coal, and the calorific value was much higher. The seams dip at a gentle angle, from 16° to 24°, and run N. 43° E., or about parallel with the Sikh-ot-alin range. But little development-work has been done as yet. There are several seams, the best one being from 2 to 3 ft. thick. The Saghalien seams occasionally are 6 ft. thick, with a firm roof and floor. It is hard to believe that so little work has been done when the conditions are so favorable.

As regards consumption, there is a fairly-large market at

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Vladivostok. More than 350 steamers arrive annually, each requiring an average of 1,000 tons of coal. The Japanese coals are inferior to the Mangugai coals, and the Chinese coal is also probably inferior. Both are too far away to compete with them. China, with six principal mines, has sold in 1910 more than 4,600,000 tons of coal, as follows:

Kaiping, near Tien-Tsin,	1,174,000 tons.
Fushun (Mukden), near Mukden,	830,000 tons.
Ping-Siang, near Kiao-Tchow,	610,000 tons.
Shan-Tung, near Kiao-Tchow,	483,000 tons.
Lanchow, near Tien-Tsin,	360,000 tons.
Pekin-Syndicate, near Tien-Tsin,	357,000 tons.

These deposits belong mainly to English and German companies, and would not affect in any way the coal-market of Vladivostok.

Oil-Deposits.

Oil has been struck on the island of Saghalien. The Tertiary formations have been penetrated under a thick cover of post-Pliocene strata which overlie transgressively the oil-formations, supposed to be Miocene.

The oil-strata ought to extend about 170 miles north and south. All the oil locations follow this line along the eastern coast, while coal is on the western coast. The direction runs parallel with the tectonic back-bone of the island, yet the locations of oil are in three different horizons, with no decision as yet as to which is the best one.

According to similar conditions in Japan, the average content of the sands would be between 30 and 50 per cent. of petroleum, perhaps higher. More prospecting-work across the post-Pliocene beds will be required in order to ascertain whether the pools will be commercially productive.

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A Trip Through Northern Korea.

BY HENRY W TURNER, LONDON, ENGLAND.

(Cleveland Meeting, October, 1912.)

THE following notes were taken on a trip through northern Korea in the fall of 1910. We started with about 19 Korean ponies, and as many Koreans, from Shin Anju, on the railway from Seoul to Antung. We passed through the ancient walled

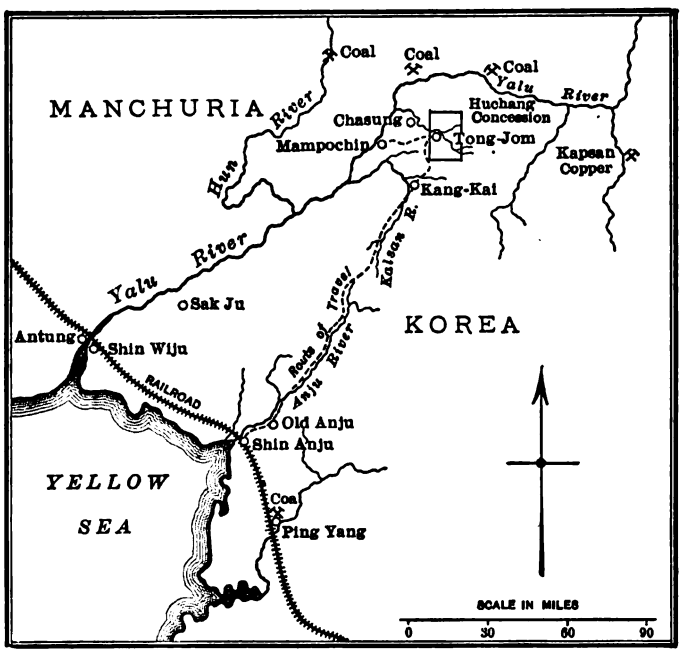

FIG. 1.—MAP OF NORTHERN KOREA, SHOWING ROUTE TRAVELED.

city of Anju, and thence up the valley of the Anju river. We then crossed over to the north into the Kalsan drainage, thence over another divide to the Chasung river, and finally reached the old Korean copper-smelting camp of Tong Jom—in all a distance by trail of about 200 miles. The return trip was made with ponies to Mampochin on the Yalu river, and thence down the river in a junk to Antung. The route of travel is shown on the sketch map, Fig. 1.

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The weather in late September and during all of October was ideal, much like Indian summer in New York State. The nights were spent in Korean houses, the method of renting them being to choose the best house and request the inhabitants to vacate, of course paying for the accommodation. These houses are fairly comfortable although of fragile construction. The floors are covered with mats and the houses are heated by means of flues under the floor. The kitchen is a separate room at one end of the house, and the heated air and smoke pass from the fire-place through the flues, making their exit by means of a wooden chimney at the other end. The floor is thus always warm when the fire-place is in operation. The same general method of heating is used in the expensive houses, as may be seen by inspecting the old palace of the former Korean emperor in Seoul. It was also used by the Romans in some of their villas in England. On the other hand, the Manchurians, living directly adjoining to the north, use an entirely different system.

The Korean ponies are thick-set, sturdy little animals, and are very satisfactory for mountain travel and for the transportation of supplies. They are usually fed with a hot, wet mixture of grain and grain stalks, and while in the stable are prevented from lying down by a band passed under the belly and fastened overhead. The males are quarrelsome with each other, but not with their riders. Each pony had a servant, or *mafoo*, to take care of it throughout the trip.

Topographical Features.

In a general way, northern Korea is a mountainous region, with deep river canyons, Figs. 2 and 6. The mountains are very irregular and no long definite chains appear to exist. The formations seem to have been subjected to an irregular uplift without great compression and folding, which has resulted in groups of hills with correspondingly irregular drainage.

Geology.

There are large areas of schistose and gneissic rocks, which seem to represent the stumps of former ranges now entirely gone or covered by more recent formations, both sedimentary and intrusive.

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The more recent (Jura-Trias?) formations make up the mountain masses of the region traversed. They have been affected by the irregular uplifts without much compression. There are thus no long definite ranges, and consequently no definite continuous zones of mineral deposits. The ancient formations, so far as seen, occupy low-lying areas, and the present mountain ranges are very likely of comparatively recent date, possibly post-Cretaceous. The uplift affected the shales which contain the coal at Pingyang and elsewhere, and the limestones and quartzites of the Huchang province. In the last named, at least, intrusions of igneous rocks accompanied the disturbances. At a still later date in Tertiary or later time there were outflows of dark basaltic lavas.

South of Pingyang along the railroad there are evenly bedded red sandstones dipping about 20°, and north of the sandstones much limestone. The shales containing the coal at Pingyang may belong to the same general series as the red sandstones.

Between Shin Anju and old Anju, all the rocks seen were schistose or gneissic and were intersected by numerous quartz-veins, and dikes of reddish feldspar, presumably orthoclase. Similar rocks, all of igneous origin, were noted along the Anju river, east of old Anju. They were succeeded by a belt of quartz-schists which dipped northerly about 30°, and in some places were folded and contorted. To the east of the quartz-schists, which represent original sandstones, we crossed a belt of clay slate and then a large area of orthoclase-porphyry cut by diorite dikes. To the east of the porphyry are white granitic rocks. In the Anju valley we saw no evidence of mineralization and no gold-placers. The only thing suggesting a metallic deposit was an old iron-smelter, indicating iron deposits in the neighborhood.

Going over from the Anju river into the Kalsan drainage, mica-schist and various granitic and gneissic rocks were seen, which continued down the Kalsan valley to within 2 miles of the old walled town of Kangkai. Then sandstones appeared, dipping NW. about 60°, succeeded near the town by shaly sediments, in places nearly horizontal.

In general the greater part of the area traversed between Shin Anju and the Chasung river is made up of igneous

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schists, gneisses, and dioritic and granitic rocks apparently barren of valuable metal deposits, and no evidence of volcanic rocks of Tertiary age was noted.

At the Chasung river, the limestone area of the Huchang copper mines begins and continues eastward to beyond the smelter town of Tong Jom. The Huchang and adjacent Hagampo copper deposits are of contact-metamorphic origin, the ore being found along the contact of limestone with intrusive quartz-diorite. The Huchang diorite area is about 9,000 ft. long and 3,000 ft. wide and is in contact with lime on the north, west, south, and southeast sides. The entire extent of this lime-diorite contact shows more or less copper, although the commercially valuable deposits are in restricted areas. On the east side of the area of diorite, where the adjoining rock is quartzite, no copper was seen. The commercially valuable deposits lie along the contact with the limestone or are associated with inclusions of limestone in the diorite. They are best developed to the north and west of Tong Jom. As is usual at such contacts, the ore consists of copper sulphides and garnet with much magnetite in spots; in other words, a mixture of primary sulphides and oxides. The sulphides identified were chalcopyrite, pyrite, and tetrahedrite or gray copper, but as at the place where the latter is found there is a fissure extending from the contact into the underlying diorite, the gray copper was perhaps deposited by solutions rising along this fissure at a time later than that of the deposition of the pyritic minerals. No distinctly secondary copper ores were noted, such as bornite or glance, yet in all probability a careful microscopic and chemical investigation would show them to be present.

In the ridge north of Tong Jom considerable development work has been done in recent years, and quite a body of 3 per cent. copper ore in a garnet gangue has been developed, as well as bunches of magnetite often containing 1 per cent. or more of copper in the form of chalcopyrite. The ore yields on assay from 30 to 50 cents per ton in gold and silver. The ore needs concentration to be of value, yet the Koreans have been smelting sorted ore here for many years, the various slag-heaps aggregating about 10,000 tons. For a mile along the limestone-diorite contact north of Tong Jom the Koreans have extracted the ore to various depths, in places as great as 100 ft. There are some fairly extensive stopes.

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FIG. 2.—TOPOGRAPHY OF KALSAN RIVER DRAINAGE.

FIG. 3.—RAPIDS ON THE YALU RIVER.

FIG. 4.—OUR JUNK ON THE YALU RIVER.

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FIG. 5.—KOREAN COPPER SMELTER, SHOWING BLOWER IN POSITION.

FIG. 6.—RIVER VALLEY NEAR SAK JU.

FIG. 7.—DIMINUTIVE GOLD STAMP MILL.

FIG. 8.—KOREAN PANNING CONCENTRATES.

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The Korean furnaces are small and of rude construction and consist briefly of a wall or back about 10 ft. high, against which is built an oven about 3 ft. in diameter. The structural material is stone and mud. The oven is filled with ore and charcoal and covered over with mud or clay, there being no chimney for the smoke and gases, which force their way out through apertures due to imperfect construction. The wall is built sufficiently high to protect the men working the blower from the heat and gases from the oven. The blower itself is of wood and hide and is shown in position in Fig. 5.

The latest igneous rocks, seen near the copper deposits, are porphyry dikes in the diorite. The gravel along the Chasung river has been extensively washed for gold, which presumably came from the copper deposits. To the southeast of Tong Jom is the Hagampo diorite area, likewise intruded into limestone. The copper contact deposits are here less extensive. Along the creek below the deposits the recent gravels have been washed for gold, which here also probably came from the copper deposits, as there are no known gold-quartz veins in the vicinity.

A few miles from the mines there are extensive wooded areas, which afford cheap fuel and timber for houses and mines. There is an abundance of water, with sufficient fall for hydro-electric works, and nearly all the little valleys and patches of soil on the sides of the ridges are under cultivation. Labor is plentiful, cheap, and good, the miners at the time of our visit receiving 25 cents per day for 10-hr. shifts.

About the middle of October, we left the copper camp for Mampochin, on the Yalu river. By the trail I noted a dark basaltic-looking Tertiary or later lava at three places, and farther west, on the south side of the Yalu, below the mouth of the Hun river, I saw some patches of black, roughly columnar rock at an elevation of about 100 ft. above the river that may represent a lava flow down the Yalu valley of a former period, presumably in late Tertiary or early Pleistocene time. So far as known, these are the only evidences of volcanic action in Tertiary or later time, thus far noted in Korea. The great Kamchatka-Japanese volcanic belt, with its accompanying siliceous gold-silver ores, lies farther to the east.

The trip down the Yalu (see Figs. 3 and 4) in a junk to

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Antung, a distance following the windings of the river of about 250 miles, was made without any incidents of note. Sometimes the junk grounded on rocks in the rapids, and it was the custom of the Koreans to set off fireworks before entering rapids to drive away the devils that are supposed to haunt them. We noted little temples or devil-houses at several places, where it was said prayers are made begging the devils to do no harm. Most of the arable land along the river is under cultivation, especially on the Manchurian side, where the tall Chinese appear to be prosperous.

The formations seen along the Yalu were in general not very different from those encountered on the trip up from Shin Anju. Granitic and dioritic rocks and granite-porphyries prevailed. Just below Mampochin on the Manchurian side are heavy bedded limestones. No evidence of mining, past or recent, was noted, but coal deposits are known north of Mampochin in Manchuria, and we saw coke made from this coal in a Chinese village in this vicinity.

We stopped at Angapo to visit the mining camp of Sak Ju, where an English company was prospecting some quartz veins which in places carry good values in gold and silver. These veins were discovered in 1906, a rather remarkable fact, as the Koreans have been mining quartz in the region for many years and the placers along the adjacent river have been worked for gold for centuries. One of the best veins was known locally as the Graveyard vein. This was in the possession of some Koreans, who were treating the ore in five diminutive 10-stamp mills on the river opposite the camp of the English company. The stamps were run by an overshot wheel, as may be seen in Fig. 7. The dies were of iron, 5.5 in. in diameter. The stamps, which weighed perhaps 50 lb., were set in wooden stems about 4 in. square. Each 10-stamp battery crushed about 1 ton of ore per day of 24 hr. The ore is first cracked up by hand to about the size of a walnut, and is fed by hand to the batteries. The mortars are of iron, about 4 in. deep, and are set into wooden sills with wooden boxes on top of the sills so as to give the mortars an effective depth of about 8 in. The actual discharge above the top of the die was in one case 5.5 in. No amalgamating plates were used but about 4 oz. of mercury per day was placed in each battery of 10 stamps, the

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mills running day and night. The pulp from the batteries flows into boxes, and the concentrates from these boxes and the amalgam and concentrates from the mortars are panned in large wooden bowls about 2 ft. in diameter and 5.5 in. deep, Fig. 8. The quartz being milled contained galena, blende, pyrite, and some pyrrhotite. It was stated that there were about 350 of these tiny stamps in operation in the district.

The gold-quartz veins at Sak Ju are in a highly metamorphosed series of siliceous sediments, soaked and intruded by a diorite-granite magma. The rocks, mica-schists and mica-quartz schists, dip southerly at low angles up to 30° with occasional higher dips. The quartz veins often follow the planes of schistosity, but fault fissure veins also exist. Some of the veins show repeated faulting. Pyrrhotite is common in the ore.

The foregoing sketch leaves the impression that northern Korea contains little to attract the mining man, but by varying the route we might have visited the camp of the Oriental Consolidated Mining Co., with a total production of gold and silver of about $16,000,000 to its credit, and likewise the Suan (*Suan*) gold mine, now being successfully operated by Collbran & Bostwick. About 70 miles east of the Tong Jom copper deposits is the Kapsan copper mine, likewise a contact-metamorphic deposit but with a good grade of ore.

The following publications give valuable information concerning the mines of northern Korea:

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Our National Resources and Our Federal Government.

BY R. W. RAYMOND, NEW YORK, N. Y.

(Cleveland Meeting, October, 1912.)

UNDER the names of Conservation, Social Justice, the New Nationalism, and Progressive Democracy, many earnest reformers are calling for a new system of Federal government to replace the one which they ascribe to our fathers, and declare to have been outgrown. They are proposing the lease by the United States of certain natural resources on the public domain, for the profit of the U. S. Treasury, instead of the sale of such resources to private citizens or associations of citizens. In the present paper, I shall try to show that this notion is neither new nor good.

It is clear that the United States is owner, as well as sovereign, of public lands acquired through conquest or purchase from other nations, as it was owner of the public lands expressly ceded to it by the original thirteen States. Its policy with regard to such lands involves, therefore, two entirely distinct questions: What shall it do as owner? and, What shall it do as sovereign? The latter question, it must be confessed, has played but a small part in our history. All the prerogatives of sovereignty which were not explicitly conferred upon the Federal government by the original States were retained by them. The ownership or control of precious or other metals or minerals in the earth, for instance, was thus reserved to the several sovereign States. New York, South Carolina, and other States have passed laws on this subject, without regard to Federal authority. As new States have been successively admitted to the Union, the tacitly accepted theory has been, that they thereby acquired all the reserved rights of the original States. These newer States largely outnumber the first thirteen; and it is inconceivable that they would submit to any deprivation of sovereignty based on historical grounds. One of them—Texas—came into the Union by its own voluntary

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act, as an organized commonwealth, already owning the whole of its territory; and Texas has always had its own land-laws, agricultural and mineral, without regard or subordination to those of the United States. In other words, the United States, not being the *owner* of lands in Texas, has not asserted, on the ground of sovereignty, any right to the ownership or control of the agricultural, mineral, or dynamic (water-power) resources of that State. Since any proposition putting the States of the Union on any other basis, and determining on historical grounds the sovereign rights of each, would be absurd, we may assume that all the States possess the same sovereign rights as Texas, and that the Federal government can constitutionally control within their borders only the administration of its own property.

This limit has been, in fact, carefully observed hitherto in Federal legislation. The U. S. "mining-laws" are not, strictly speaking, mining-laws at all. They are simply statutes regulating the disposal of mineral lands belonging to the Federal government; and they leave to local authority the regulation of the industry of mining itself. All put together, they affect nothing but the title derived by the miner, as tenant or purchaser, from the United States as owner.

With regard to the mineral resources owned by it in existing States, or in Territories which are destined to become States, the policy of our government hitherto has been, since 1866, to treat such natural resources as the prospective property of the citizens of the States within which they are located, subject to the regulation, control, and taxation by those States; and the lands containing such resources have been either conveyed outright to the States concerned, or sold to individuals, to be thereafter controlled, as to their development, by the local governments.

The first public domain of the United States was acquired by cessions from some of the original thirteen States, covering what is now Tennessee, Illinois, Indiana, Ohio, Michigan, Wisconsin, that part of Alabama and Mississippi lying north of the 31st degree of North Latitude, and that part of Minnesota lying east of the Mississippi. Nearly all the public land in Tennessee was absorbed by the claims of North Carolina, and the remainder was subsequently ceded by the United States to Tennessee.

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The rest of the territory granted by the States to the Federal government included our first public domain, together with lands already owned by private persons.

The first Federal legislation concerning the mineral resources of the public domain is found in the famous ordinance of 1787, concerning the Northwest Territory (comprising what is now Ohio, Illinois, Indiana, Michigan, and Wisconsin), which provided that one-seventh of the land should be given to the soldiers of the Continental Army, one-seventh reserved to the United States, one-thirty-sixth set apart for the support of schools, and the rest allotted to the States "*pro rata*," and sold at auction by the State Commissioners, who were to give deeds, excepting and reserving to the United States one-third of all gold-, silver-, lead-, and copper-mines. The same reservation applied to the bounty- and school-lands also.

I do not propose to trace here the further course of this legislation, or the history of the Northwest Territory, out of which Ohio was carved in 1800, Michigan Territory in 1805, and Illinois Territory in 1809, leaving (under the name of the Indiana Territory) what became in 1816 the State of Indiana. Nor is it necessary to discuss here the Louisiana purchase of 1803, or the question whether Idaho, Washington, and Oregon were acquired by that bargain, or belong to the United States by virtue of the discovery of the Columbia river by Capt. Gray (1792), the exploration of Lewis and Clarke (1805), the settlement of Astoria (1811), or the cession of "the Floridas" by Spain (1819).

In 1807, Congress enacted that the lead-mines in the Indiana Territory (then comprising Indiana and Illinois) should be "reserved for the future disposal of the United States;" that any grant by the United States of a tract in which a lead-mine had been previously discovered should be considered fraudulent and void, and that the President was authorized to lease for not more than five years any lead-mine which had been or might thereafter be discovered in the said Territory.

This act inaugurated the policy of leasing the public mineral lands. The leases executed under it covered tracts, at first 3 miles and afterwards 1 mile square, and required the working of the mines with due diligence, and the return to the United States of 6 per cent. of the ore mined.

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This policy was continued for 40 years, uninterrupted by any further legislation, except the Act of June 26, 1834, creating an additional land-district in Wisconsin, which authorized the President to offer for sale certain public lands, "any law of Congress heretofore existing to the contrary notwithstanding." Many Acts between 1807 and 1834 had dealt with the pre-emption and disposal of public lands; but nearly all, if not all, of them had continued the reservation of 1807 as to lead-mines. But this Act of 1834, expressly nullifying previous reservations, seemed to permit the pre-emption and acquisition of the lands which it specified, whether they contained lead-mines or not. At all events, this was the plea of the defendant in the case of U. S. *vs.* Gear.[1] In that case, decided in 1845, the U. S. Supreme Court held defendant guilty of trespass, on the ground that the reservation not expressly contained in the Act of 1834 was to be regarded as a part of it, because it had been for many years the consistent policy of the United States, and could not be set aside by inference from the terms of a law, the declared purpose of which was simply the creation of a new land-district. In a previous case,[2] the Court had said in 1840:

"It has been the policy of the government, at all times, in disposing of public lands, to reserve the mines for the use of the United States."

The decision in the Gear case was reaffirmed in Cotton *vs.* U. S. (11 Howard, 229), which applied the same principle to the cutting of timber upon public land, declaring that, as the owner of the land, the government had the same rights as an individual owner to protect its property.

The constitutionality and equity of these decisions cannot be seriously questioned. They rest upon the ownership by the United States of the lands concerned, and therefore upon the common-law right of a holder of the complete title—not upon the prerogatives of a sovereign. Yet there is, as Curtis H. Lindley says, no doubt that the reservation of part of the product of certain kinds of mines—for instance, in the Ordinance of 1787, above cited—was suggested by the similar provisions in the Crown grants, upon which the titles of the original States were based. We do not find, however, that these *regalia*, or

[1] 3 Howard, 120.

[2] U. S. *vs.* Gratiot, 14 Peters, 538.

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rights of the sovereign, as distinguished from the rights of the land-owner, were ever seriously enforced. The policy of the government, so plainly adopted in 1787 and 1807, was applied to its own lands only.

It is interesting to note in passing that the doctrine of sovereign rights over mines has been at least twice asserted by single States against the United States, and not without some appearance of logical force. For, if the United States, being at once a land-owner and a sovereign, does not possess, or has waived or ceded, its claims in the latter capacity, then, in the former capacity it can be dealt with by the sovereign State, like any other private land-owner. Thus it was held, in 1853, by the courts of California,[3] that the State, by virtue of its sovereignty, enjoyed all the *jura regalia* belonging, under the common-law, to the King of England. The doctrine appears to have been that, whoever owned the land containing the mines, the Sovereign only could control their operation. In the case of California, such a sovereignty could not have come from the King of England and must have come through the King of Spain and his assign, the Republic of Mexico, the rights of which had passed, through conquest, to the United States. The only ground of California's claim must therefore have been, that by its admission to the Union it became a sovereign State, clothed (from whatever source) with the rights enjoyed by the original thirteen. Whatever merit this claim may have had, it was subsequently abandoned by the California courts.[4]

But a similar doctrine was apparently followed by the Legislature of Michigan, in its Act of April 28, 1846, which declared that property in all mines of gold or silver, or of other minerals containing gold in any proportion, is vested in the people of the State of Michigan by virtue of their right of sovereignty. Since the Federal land-laws applied to public lands in Michigan until 1873, this assertion of "sovereignty" over certain mines, if extended over such lands, could be based only on the theory that the sovereign rights of the original States which ceded to the United States the territory out of which the State of

[3] Hicks *vs.* Bell, 3 Cal., 219. I take this statement and reference from Curtis H. Lindley's admirable and almost exhaustive treatise on American Mining Law, 2d ed., San Francisco, vol. i., Sec. 21, p. 42 (1903).

[4] Moore *vs.* Snow (1861), 17 Cal., 199, 217; and later cases cited by Lindley.

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Michigan was subsequently segregated were somehow held in trust by the United States, and afterwards tacitly re-conveyed to the State at the time of its admission to the Union. No conflict having arisen under this claim between the State and the Federal authorities, it may be regarded as practically settled that neither party now asserts "sovereign," as distinguished from proprietary, rights to mines.

The policy established in 1807, of reserving from sale, and holding and leasing, the public mineral lands was, as I have said, maintained for 40 years—a considerable period, in view of the total length of our country's history.

In December, 1845, President Polk, in his first annual message to Congress, reported that the results of the leasing of mineral lands had been in every respect unsatisfactory. The revenue to the government had been less than one-fourth the expense of administration; litigation and wasteful mining had been promoted, etc.; and he recommended the abandonment of the system. In 1847, this recommendation was adopted by Congress, and the long experiment of leasing public mineral lands came to an end—not to be revived until, after more than 60 years, it was proposed as a new idea by a generation which had forgotten the experience of the fathers.

The discovery of gold in California, and the consequent rush of many thousand adventurers into the unsettled and unsurveyed territory acquired from Mexico by the treaty of Guadalupe Hidalgo, put a new face upon the question of Federal ownership and sovereignty. The pioneers who thronged to the gold-bearing gulches not covered by any asserted private ownership were clearly, according to the previous decisions of our Supreme Court, trespassers upon the property of the United States; and, in all conflicts of title among them, the superior title of the United States would theoretically operate as a bar to such "equity" remedies as writs of injunction or ejectment. But the greater part of the disputes over mining-titles in those regions involved just such remedies; and it was difficult to see, where both parties were trespassers, how one could ask protection against the other.

The courts solved this difficulty in a summary and sensible way, by leaving out all consideration of the superior title of the United States, and adjudicating the relative rights of possessory

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claimants to public mineral land, with the tacit proviso that the rights thus declared were subject to any future action on the part of the real owner. Under this *modus vivendi*, the treasures of the Comstock lode and of many other rich deposits were claimed, fought for, and won by individuals or corporations which had no original title to them, but only, according to local regulations, a better claim than their antagonists. Yet these regulations were made by communities of trespassers, without regard to the superior authority of the Federal landlord.

This curious situation lasted until 1866, when Congress passed an Act, declaring the occupancy and exploration of the public mineral lands free to all citizens, and confirming the rights which had been acquired in these lands under local regulations. In other words, the United States ratified whatever had been legally done without its assent, and agreed that its silence should be, within certain limits, interpreted as consent. This Act was confessedly crude and imperfect, and it was found to be particularly objectionable, in that it did not provide, either through possessory occupation and development, or through purchase and patent, for the protection of the occupant of a piece of mineral land (whether as tenant or as grantee of the United States) against predatory explorations and exploitations of his location. It was "the lode" only, with its "dips, spurs, and angles," which was granted under this law. And, since the law recognized the conditions of title imposed by "local regulations," and the one universal condition of such regulations was priority of discovery and location, this became naturally the basis of possessory title.

It was not until 1872 that "The Law of the Apex" came. This was a crude attempt to remedy the defect of the Act of 1866, by protecting the occupants of mining-claims against piratical trespass, through the grant to them of a right to the land, as well as the "extralateral" right. Unfortunately, these rights were made dependent upon a new and previously unknown condition—the "apex"—and subject to the similar rights of any neighboring "apex."

It is unnecessary to dwell here on the positive and negative defects of this law. It was a great mistake to make mining-titles dependent upon future geological developments. The statute created greater evils than those it was designed to

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cure. And, in my judgment, it will never be too late to repeal it, and to substitute the simple and logical system followed by all other nations and by the majority of the States of the Union.

Nevertheless, the Acts of 1866 and 1872, however defective and mischievous they may have been, possessed one feature, which their faults did not nullify: namely, they invited the exploration and occupation of the mineral public lands, and thereby stimulated a development unparalleled for rapidity and extent. This result would have been equally remarkable, and still more satisfactory, if the liberal policy of the United States had been so exercised as to give safe and definite titles to the occupants and purchasers of its mineral lands. The circumstance that a tenant or grantee of the government was liable to an interminable risk of losing title, by reason of geological developments concerning the "apex" of his lode, would not seem, at first glance, to be an additional encouragement to his enterprise. Still, it must be confessed that, during the period between 1866 and 1872, and for some years after the latter date, the "extralateral right" attached to a mining-location was so attractive to prospectors and possessory owners as to counterbalance the two great statutory evils—namely, during the first period, the total lack of protection of the claim-owner against trespass upon his claim, and during the second period, the uncertainty of title caused by basing all the locator's rights upon the location and course of the "apex" of the located lode, without regard to the first discovery thereof. The positive grant of the "extralateral right" loomed large in the eyes of the prospector and his assignees; while the precarious conditions of that grant, and the express reservation of the corresponding "extralateral right" of a neighbor,—raising questions which could only be settled by litigation, future development, expert evidence, etc., during which the United States would simply stand back, and let its grantees fight out for themselves the exact meaning of its grant,—were not realized at once as real defects of title. These defects in our mining-law have cost American citizens many million dollars. Yet, in spite of this huge, unnecessary handicap (imposed without the least benefit to the government), the policy of transferring the mineral public land to individual ownership upon terms so

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liberal as to amount practically to a gift, has resulted in the creation of a new empire, west of the Rocky mountains, with a rapidity unprecedented in human history. I think no one would venture to dispute the proposition, that, through its homestead-law, its mineral-land laws, and its grants in aid of transcontinental railroads, the United States has built up a new, rich, loyal Western realm, which could never (or only after much longer processes) have been established in any other way.

It is now proposed to abandon the policy which has produced such results, and to revert to the plan which so conspicuously failed a century ago. The reasons for that failure still exist. The causes which produced confusion, corruption, and waste before are even more likely to produce them to-day. It is not easier to administer the mineral deposits of Alaska than it was to manage those of Illinois. It remains as true as ever, that the wealth, prosperity, and loyal union of our States is best served by placing their natural resources, as rapidly as possible, in the hands of their citizens, subject to local taxation and regulation. We have discovered no new principle in democracy which nullifies Spencer's famous declaration that a democratic is the worst form of government for doing those things which no government ought to do.

The following propositions concerning the proposed policy are submitted with considerable confidence:

1. It is never desirable, even though it be sometimes necessary, that private enterprise should be subject to governmental interference or competition.

2. If any government in this country must thus interfere, it should be the local (municipal or State), and not the Federal, government. Among the many arguments for this proposition I may mention a few.

a. As a matter of statesmanship, the Federal government cannot afford to put one State (or Territory destined to become a State) on a different basis from others; and since it cannot legally administer the natural resources of New York or Texas, the attempt to do so in Idaho or Alaska would be unwise. This point I have already discussed.

b. It is a well-known teaching of experience that the government dealing with local interests and industries should be, if

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possible, a local government, easily watched and reached by the citizens affected. In my discussion of Mr. Bain's paper on Alaska Coal-Land Problems,[5] I have emphasized this point, and will not here repeat what I said there. But I may add that a central government attempting to perform such local supervision invariably fails to do that work well, and incidentally falls short of vigilance and efficiency in the functions more properly belonging to it. Take, for instance, the administration by the British Parliament of the internal affairs of Ireland, Scotland, Wales, and England, as well as India. The case is plain: Parliament has too much to do, and consequently delegates to committees and departments the work for which it is nominally responsible. Another instance may be found in the government by Congress of the District of Columbia, which receives perfunctory attention at a couple of sessions during the whole term. The necessary result is that the affairs of the District do not receive the vigilant attention and public discussion which they would get from a District legislature, not burdened with the management of national, international, and colonial questions, and the manufacture of party ammunition for presidential campaigns. Moreover, the actual and necessary business of the national government is neglected by those executive departments which are loaded with extra-constitutional duties. Administrations not seldom claim much credit for the fearless exposure of long-continued department abuses which, in fact, they could have prevented if they had not been so much occupied with other people's business that they neglected their own.

3. Of all the methods of governmental administration of natural resources, that of leasing, as now widely advocated, is the worst. I refer to the system of short leases (the term usually mentioned being 25 years), coupled with the requirement of continuous operation. I could ask no better illustration than the history of the Pennsylvania anthracite-region. In his paper on Alaska Coal-Land Problems,[6] Mr. Bain gives a summary of this history, which candidly confesses the beneficial effects wrought by the large corporations, so often ignorantly denounced. But he omits to mention the greatest cause of the

[5] *Trans.*, xliii., 595 (1912).

[6] *Idem.*

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confusion and waste which those corporations reformed, namely, the system of short leases prevalent throughout the anthracite-region. No amount of stipulation, regulation, and inspection can make it the interest of a lessee to spend money on improvements for the benefit of his successor; to put up, at extra cost, machinery and buildings for permanent use, after his term; or to sacrifice immediate profit in favor of technical efficiency and economy. And no amount of governmental pressure can be reasonably or practically applied which will make a mine-tenant do as much for conservation, scientific improvement, etc., as he would voluntarily do if he were the owner of the mine.

This is peculiarly true of coal-mines. I notice that the advocates of a leasing-system always propose to bind the lessee to operate continuously; yet this condition is fatal to economy. Mr. Parker, who has observed the evils of hot competition in the bituminous coal-mines, thinks the government should interfere by ordering those mines to be shut down which do not pay![7] But why do they go on, if they do not pay? Doubtless because their owners would rather sacrifice some money for a while than incur a greater loss by stopping short. All sorts of reasons make a man go on for a while in a losing business; and operators possessing large capital can often afford to do so, for the sake of a more profitable future. But to make the title to the property depend on its continuous operation is to put large and small capital into common ruin, and to ordain deliberately the waste of natural resources. If government leases were granted for very long terms, and (like many private mine-leases) could be held by annual payments during periods when the mine was not worked, this objection would be measurably met. But that would be practically granting ownership of the mines—which is what wisdom and experience recommend, and not what our reformers want.

4. Why do we not hear of the leasing of gold-, silver-, and copper-mines, and of agricultural lands? The argument for these applications of the system would be equally strong. I fear the reason for its conspicuous absence is, that coal-mines require capital, and that capital is an easy mark for destructive attack. But, as has often been observed, capital does not fight, except when cornered. As a rule, it runs away.

[7] The Conservation of Coal in the United States, *Trans.*, xl., 601 (1909).

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This, I think, will be the probable end of the cycle into which we are entering. If we try this new system, which is not new, we shall find, as was found three generations ago, when it *was* new, that it is not good; and we may then be content to try once more the old-fashioned road on which our nation traveled to greatness. The men who laid out that road "did not foresee," we are often told, the development of the vast region of the future through which it was to lead. But they were skillful surveyors; they knew where the road should be placed;—and historians tell us that roads are the most permanent works of man!

Discussion.

George Otis Smith,* Washington, D. C. (Presented in oral abstract by Edward W. Parker):—Dr. Raymond, in his paper, discusses a public question the consideration of which is both timely and appropriate. To his review of the historical and legal phases he brings an exceptional fund of experience and observation, and even were it necessary it might seem presuming to venture upon a discussion of this portion of his thoughtful paper. However, modern problems must be solved on the basis of modern conditions, and finding myself compelled to differ with the distinguished dean of American mining engineers, I shall confine my comments to the four propositions submitted by Dr. Raymond.

The first of these is this:

"It is never desirable, even though it be sometimes necessary, that private enterprise should be subject to governmental interference or competition."

This fundamental premise, while not discussed by Dr. Raymond, is surely open to debate. Whatever is "necessary" is surely to that extent "desirable" even if not "desired" by all concerned. On broad grounds, it is plain in this day and generation that "private enterprise" cannot hold itself aloof from governmental regulation, and what some of us perhaps regard as regulation, others may term "interference or competition." The individual and the community are in business partnership, and as Dr. Butler has said "both partners must be heard in respect to the policies which the partnership wishes to pursue." Individual initiative, with self-interest as its impelling force, must be kept subordinated to the common

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good, and in the evolution of popular government this control has become not less but more effective. Until we have become a nation of what Governor Marshall has termed "automatic citizens"—who realize "that they may 'go the limit' in the way of success, but that they must not injure their fellowmen"—it will continue to be necessary for government to regulate private enterprise.

Dr. Raymond's second proposition is this:

"If any government in this country must thus interfere, it should be the local (municipal or State), and not the Federal, government. Among the many arguments for this proposition I may mention a few.

"*a.* As a matter of statesmanship, the Federal government cannot afford to put one State (or Territory destined to become a State) on a different basis from others; and since it cannot legally administer the natural resources of New York or Texas, the attempt to do so in Idaho or Alaska would be unwise."

The evident tendency in the last half-century has been to strengthen Federal powers and to expand Federal functions. This movement is towards the simple and the uniform, since heterogeneity in State legislation in matters that are nation-wide leads to abuses, as, for example, divorce laws and corporation laws. But the obstacles that may hinder progress in the enactment of many Federal laws need not be encountered in the matter under discussion. As Dr. Raymond has suggested in his title, it is National Resources that we propose to utilize and develop, and for the Federal authority to retain its present jurisdiction over the mineral wealth in the public lands will, therefore, not prevent confusion of law, but it will at least provide for uniform action over the large remainder of this class of national resources. To illustrate: the economic development of potash-deposits in Nevada or California may be of more vital concern to the Southern and Mississippi Valley States than to either Nevada or California. The natural resource is a national resource.

Dr. Raymond gives as a second reason for local rather than Federal control that

"It is a well-known teaching of experience that the government dealing with local interests and industries should be, if possible, a local government, easily watched and reached by the citizens affected."

I can but think that Dr. Raymond has misinterpreted public opinion, and has read erroneously the signs of the times

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in making this statement. I should say rather that the teaching of experience informs us that government to be effective and impartial, at least where highly-valuable privileges that stir the cupidity of men are involved, should be, not a local government created and controlled by those most directly interested, but a government sufficiently distant to be able to deal calmly and firmly with all types of questions which may arise, and especially with those questions in which the interests of the people of the United States as a whole may clash with those of the locality immediately concerned, or in which future and present advantages appear to conflict. Legislation upon which administration and the action of the courts depend, is broader and freer from the influences of powerful local interests if it be Federal, than if it be State legislation. In *The American Commonwealth* so disinterested an observer as James Bryce sets forth his opinion on this point in the following terms:

"As the frame of a State Government generally resembles the National Government, so a State Legislature resembles Congress. In most States it exaggerates the characteristic defects of Congress. It has fewer able and high-minded men among its members. It has less of recognized leadership. It is surrounded by temptations relatively greater. It is guarded by a less watchful and less interested public opinion."

The third of Dr. Raymond's propositions is epitomized in the following sentence:

"Of all the methods of governmental administration of natural resources, that of leasing, as now widely advocated, is the worst."

While so far as government lands in the United States are concerned, experience in leasing is limited chiefly to Indian lands and to isolated cases like that of the Owl Creek Coal Co., now operating under the authority of Senate Joint Resolution No. 100, approved Aug. 24, 1912, there is every reason to believe that with the granting of proper legislative authority, the system can be made just as successful as it now is in the States of Colorado and Wyoming, in the Australian commonwealths, and in certain parts of the Canadian Dominion, for leasing *per se* is anything but an experiment. It is a matter of common knowledge that a large proportion of the coal-mines of the East are operated on the leasehold basis. To be sure, the landlord is a private land-owner, either individual

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or corporate, instead of the government, and the royalties charged are higher than any that have been contemplated by the government. Nonetheless, leasing is a success even under these adverse conditions. Wyoming coal-lands have been under lease from the State since 1907 and those of Colorado for a much longer period. The income from royalties on State coal-lands in Colorado has nearly doubled in each successive biennial period from 1902 to 1908. In the Australian States the system was deliberately adopted after a comparison of the leasehold tenure and the fee tenure, in at least one of the States—New South Wales—the systems operating for a period of 30 years side by side. After this deliberate and thorough trial the system of disposing of mineral lands in fee has been abandoned and the leasehold system adopted exclusively.

Aside from the belief that a system of government leasing can be made entirely successful from the point of view of the operator as well as from that of the government, there are many general considerations in its favor.

A sale in fee of the government's rights to its mineral resources places forever beyond the control of the people elements which may be vital to their welfare. It is a final verdict rendered now as to all the future. It is a step which cannot be retraced, no matter what emergencies may arise hereafter. Leasing, on the other hand, means that the nation does not finally part with its mineral deposits except as they are extracted and used, and that the terms on which these resources may be developed or sold will be subject to readjustment as conditions may change. Such a power to readjust is essential, since no man can foresee how vital any single element may become.

Leasing is fairer to the operator than purchase. Under the leasing system he pays for his product as he develops it and pays for the precise quantity that is developed, and for that only. Under any form of purchase a fixed sum is paid prior to development and before the quantity of the resource purchased is accurately determined. When sales are made by the government the doubts as to quantity are resolved in favor of the purchaser, and it is probable that he very rarely receives less than he pays for; but even here it may sometimes happen that barren ground is bought and the amount expended for it is

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wasted. If, on the other hand, purchases are made from private owners, who may be simply middlemen, who acquired title from the government, no effort is made to protect the operator by resolving doubts in his favor, and he may not only buy unproductive ground, but may pay an excessive price for even that which shall prove to be productive. The penalty for these excessive prices is usually eventually paid by the consumer. The operator who has made an unwise purchase of coal-lands will recoup if possible by charging a high price for the coal that he recovers.

The purchase system lends itself readily and directly to a condition of monopoly. The acquisition of large bodies of mineral land is not within the power of even the moderately well-to-do. It is, however, within the means of the wealthy group or the powerful corporation. A familiar method of suppressing competition is through this acquisition of the sources of supply. Such sources once acquired may not be used for a generation or longer, but they are withheld from other users, this being one of the objects of their purchase. Leaseholds operated under government supervision, subject to proper safeguards as to transfer and to sufficiently short periods for the revision of terms, would not lend themselves so readily to monopolistic control. Furthermore, when purchases are made and the property is held for long periods, interest charges accumulate and a resulting higher price must be paid by the final purchaser of the product. Since the lessee is not a purchaser he has no interest to pay on a purchase price and, therefore, need not secure a return of this interest in his charge to consumers.

The ownership of mineral land, even by the wealthy, tends constantly to force untimely development. Capital thus tied up seeks release. Its owners will sell or operate the moment a satisfactory price is in sight, even though the general condition of trade does not warrant it. Under the leasehold system there is less of this constant pressure of locked-up capital to escape. The result must be a freer adjustment of production to demand and a healthier tone throughout the trade.

Opinion among high officials, recognizing the force of these arguments, is swinging towards the leasing policy as applied to certain, at least, of the nation's mineral resources.

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The President, in an address delivered before the National Conservation Congress in St. Paul, Minn., on Sept. 5, 1910, expressed himself favorably upon the question of leasing certain of the minerals in lands still in public ownership. As to coal-lands he said:

"If the Government leases the coal lands and acts as any other landlord would, and imposes conditions in its lease like those which are now imposed by the owners in fee of coal mines in the various coal regions of the east, then it would retain over the disposition of the coal deposits a choice as to the assignee of the lease, a power of resuming possession at the end of the term of lease, or of readjusting terms at fixed periods of the lease, which might easily be framed to enable it to exercise a limited but effective control in the disposition and sale of the coal to the public. . . . As one-third of all the coal supply is held by the Government, it seems wise that it should retain such control over the mining and sale as the relation of lessor to lessee furnishes."

Speaking of the public oil- and gas-lands, the President expressed himself as follows:

"The needed oil and gas law is essentially a leasing law. In their natural occurrence oil and gas cannot be measured in terms of acres like coal, and it follows that exclusive title to these products can normally be secured only after they reach the surface. Oil should be disposed of as a commodity in terms of barrels of transportable product rather than in acres of real estate. This is, of course, the reason for the practically universal adoption of the leasing system wherever oil land is in private ownership. The Government thus would not be entering on an experiment, but simply putting into effect a plan successfully operated in private contracts. Why should not the Government as a land owner deal directly with the oil producer rather than through the intervention of middle men to whom the Government gives title to the land?"

"*The principal underlying feature of such legislation should be the exercise of a beneficial control rather than the collection of revenue.* . . . Much more important than revenue is the enforcement of regulations to conserve the public interest so that the covenants of the lessees shall specifically safeguard oil fields against the penalties from careless drillings and of production in excess of transportation facilities or of market requirements."

In discussing the phosphate-bearing public lands the President made this statement:

"A law that would provide a leasing system for the phosphate deposits together with a provision for the separation of the surface and mineral rights, as is already provided for in the case of coal, would seem to meet the need of promoting the development of these deposits and their utilization in the agricultural lands of the west."

Secretary Fisher, upon his return from a tour through a portion of the coal-fields of Alaska in the summer of 1911, deliv-

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ered an address upon Alaskan problems before the American Mining Congress at Chicago, Ill., on Oct. 27, 1911. In this address he expressed his conclusions as to the proper disposition of the coal in the Alaskan fields, incidentally stating some general principles which he felt should underlie any governmental policy that may be adopted. I quote as follows:

"I believe that the time has passed when the Government should convey an unrestricted title to its coal-fields. The day is done in which the Government should deliberately encourage the unrestricted private exploitation of the sources of power. To impose effective regulations upon these sources after they have passed to private individuals in fee is exceedingly difficult, even if not impossible. The ownership of the fee carries with it under the law the right of unrestricted sale, and many regulations which are desirable in the public interest can be imposed, if at all, only after radical changes in the laws and prolonged litigation in the courts, perhaps only after Constitutional amendments. It is therefore unwise, and in my opinion unnecessary, to sell our coal-lands in order to secure their effective development."

He states further:

"I believe that the leasing system avoids the controversies and the difficulties of both extremes of public and of private ownership. It has been adopted with conspicuous success in the great mining communities of Australia and New Zealand. It is now the established law of the Yukon Territory, lying in Canada just across the border line from Alaska. It is the system under which much of the privately-owned coal-land of the United States is in fact to-day being developed. Under it we can insert as matters of contract and as conditions to which the lessee voluntarily consents those regulations and requirements which promote the public interest, the enforcement of some of which by mandatory law might be unconstitutional. By making the details of our leases liberal we can make them seem more attractive to capital than if we adopt the policy of an outright sale of the fee."

A leasing law is the almost universal form of tenure adopted in the Australasian mining States. Its adoption there has been the result of gradual evolution from a system of fee tenure inherited by these Anglo-Saxon peoples from their English ancestors. The leasing system is therefore not an accident in the Australian States. It represents the deliberate judgment of mining-men and law-makers. The attitude there may be illustrated by the following quotations from the report of Special Commissioner Veatch to President Roosevelt as a result of investigations made during the winter of 1907–08. Reporting upon the leasing system in the Commonwealth of Victoria, he says:

"The Royal Commission on Gold Mining, appointed in 1899 to inquire, among other things, into the best method of promoting mine development in Victoria,

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examined about 500 witnesses and found no man who recommended freehold. This Commission, including prominent mining men, capitalists, and members of Congress, instead of recommending freehold as a means to this end, states: 'The great bulk of the evidence throughout the whole course of the inquiry bears out the opinion that no greater mistake can be made by the State than that of alienating mining lands. There is no necessity whatever for disposing of such areas in fee simple. No use to which the land can be applied requires that it should be sold.'

"This recommendation is made in a country which entirely repudiated the doctrine of the nationalization of the land in an agricultural sense, and by some of the very men who have been prominent in this repudiation."

The experience of these practical pastoral and mining peoples, then, leads them to the advocacy of the adoption of a leasing system for mineral lands and the retention of the fee system for agricultural lands. This chapter from experience gained in our southernmost continent is at least a partial answer to the argument so often made, and repeated by Dr. Raymond as a part of his fourth proposition, to the effect that if the leasing policy is adopted at all it should be extended to the public agricultural lands as well as to the mineral lands. One more quotation may be made from Veatch's report to indicate the attitude of another Australian State, namely, New South Wales, as a result of many years' experience with both systems of land tenure:

"New South Wales, after fifty years of indiscriminate disposal of minerals and thirty years of a system in which mineral alienation and Government leasehold were tried side by side, and in which mining men showed a marked desire for a leasehold tenure, adopted Government leasehold as the most satisfactory method of dealing with minerals. This last policy has been tried exclusively for almost twenty-five years, and has been found to be so satisfactory that there is not even a remote possibility of the plan being changed."

Dr. Raymond, like other opponents of the leasing system, cites the law of 1807 under which lead-mines of the Mississippi valley were made subject to lease, and so remained until Congress, in 1847, acting upon the recommendation of President Polk, repealed the law, whose operation had been unsatisfactory. Secretary Fisher, in the address heretofore quoted, meets the position of those who consider the experience gained in the period from 1807 to 1847 as settling adversely once and for all the feasibility of operating mining-property under leasehold tenure in the United States. He says, and his words might well have been written in anticipation of Dr. Raymond's address:

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"The opponents of the leasing system delight to dwell upon the fact that in the first half of the last century the Federal Government undertook to lease the lead-mines on the public domain, and that the effort was not successful and was abandoned in 1847; but when we consider the conditions under which the attempt was made, and especially that the leases were limited to five years, with a royalty of one-sixth of the lead for Government use; that the validity of the lease was constantly attacked upon the ground that the statute did not contain the necessary provisions for carrying it into effect; and that the system was permeated with the same kind of fraud and evasion which until recently characterized the operation of certain of our later land and mineral laws, we can understand the failure, and that the experiment is of no value whatever in determining the merits or demerits of a properly drawn and properly enforced law for the leasing of the public coal lands of to-day."

In summary, it may be said that the advocates of the leasing system believe: (*a*) That it represents much more statesmanlike disposal of our mineral resources than the sale in fee, because it permits future generations to adapt their policies to the conditions of that time. Final alienation of the public interest in its mineral deposits by granting titles in fee to these deposits removes forever the possibility of control by the nation in its own interest, and fixes upon the country for all time the right of private and corporate owners to exercise this control in their own interests; (*b*) that the leasehold system is fairer to the mining-operator than the system of fee titles, because under it he pays for precisely the quantity of the mineral product which he mines and markets, and not for some estimated but unmeasured quantity. He is relieved of the necessity of tying up capital by purchase of mineral lands for future development in order either to secure an improper monopoly, or to properly protect himself from successful competition. It is better for the consumer because the disadvantageous elements from the operator's point of view must be met by him through the imposition of increased charges to be paid eventually by the consumer. It should be better for both the operator and the consumer, because it would tend to eliminate the non-productive middleman, who now in the majority of cases interposes between the government and the operator, and imposes upon the latter, and through him upon the consumer, a speculative charge, in many instances very heavy, that tends both to reduce the operator's profit, and to increase the charge to the consumer; (*c*) a Federal leasehold system will result in uniform laws, and uniform administration of those laws upon the public mineral lands to which they are applied. If a leasing law

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is not enacted, and the mineral lands are disposed of as heretofore to private owners, they will then become subject to the various State laws with all of their heterogeneity; (*d*) government leasehold systems are not experiments. They are in successful operation over widely-separated sections of the globe. Where fully and freely tried they meet with favor from miners and consumers alike, and there is no tendency to revert to the fee system of disposal.

All of us who are interested in the manifold problems involving the administration of the public lands welcome the active discussion of the leasing system that is going on at present, and we particularly welcome thoughtful and dispassionate contributions to this discussion by students of mining-problems of the experience of Dr. Raymond. The final outcome must be the wisest solution of the problems considered, and that outcome, whether it meets our individual views or not, will be accepted and the laws that represent its expression will be loyally supported and administered by all citizens.

In conclusion, I should like to correct what I am sure is an inadvertence on Dr. Raymond's part. He appears to assume that all who advocate a leasing system couple with it "the requirement of continuous operation." In my annual report for 1911 considerable space was given to mineral-land legislation, and under the subject of needed oil-land laws, I distinctly stated that continuous operation should not be required under leasehold, in the following sentences:

"The chief advantage of the leasehold for oil over a fee-simple title lies in the prevention of monopolization through large holdings. Such large holdings without production would be guarded against by a ground rental sufficiently high to discourage the *acquisition* of lands except for immediate and continued development. . . . *This indirect control of development would be preferable to the direct enforcement, by forfeiture, of continuous production, which should be avoided because of the danger of disturbing the delicate equilibrium between supply and demand.*"

Dr. Raymond states near the conclusion of his address that Mr. Parker, of the Geological Survey, "who has observed the evils of hot competition in the bituminous coal-mines, thinks the government should interfere by ordering those mines to be shut down which do not pay."

What Mr. Parker actually said differed very essentially from that which Dr. Raymond implies that he said. I quote a paragraph from his paper on The Conservation of Coal in the United States, delivered before the Spokane meeting of the Institute in September, 1909:

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"Under .... government control bituminous-coal mining could be regulated through a system of license; and in order that restriction on coal-production may be secured, no license should issue for the opening of a new mine until ample proof is shown that the necessities of the people or of trade require it."

What Mr. Parker actually advocates, then, is not that operating mines should be shut down, but that new mines should not be encouraged to open until there shall be plain need of their product. The policy actually advocated is one of the conservation of capital. The policy imputed to Mr. Parker was one of destruction and waste of the capital. The difference is wide.

R. B. BRINSMADE, Puebla, Puebla, Mexico (communication to the Secretary *):—Though I heartily approve of several minor points emphasized by Dr. Raymond (such as the general desirability of local administration, rather than Federal, and the evils inseparable from the Federal Apex Law), I feel obliged to dissent from his four final propositions.

As Dr. G. O. Smith has already discussed these propositions from the standpoint of the experience of other countries, I shall base my criticism upon certain general economic principles, which will be grouped under the following five topics:

1. The dual theory of property.
2. The essential factors in fixing prices.
3. The relation of land ownership to the distribution of wealth.
4. The relation of land ownership to speculation and development.
5. The relation of land ownership to the political power.

I shall give here only a brief outline of these topics, as they are fully treated in the standard treatises on political economy.[1]

1. *The Dual Theory of Property.*

This theory divides all property into two distinct classes: land, and labor product. By land is meant the earth, and it includes all natural resources; its value is calculated after deducting all improvements due to labor and it includes five main classes: viz., farms, forests, mines, town sites, and public-utility

* Received Mar. 15, 1913.

[1] *Wealth of Nations*, by Adam Smith, 11th ed. (1805).
Political Economy, by David Ricardo, 3d ed. (1821).
Political Economy, by John Stuart Mill, 6th ed. (1865).
Political Economy, by J. E. Symes, 5th [ed. (1904).
Political Economy, by Charles Gide, 2d Amer. ed. (1904).
Progress and Poverty, by Henry George (1879).
Capital, by Carl Marx, vol. iii., part 6 (1894).

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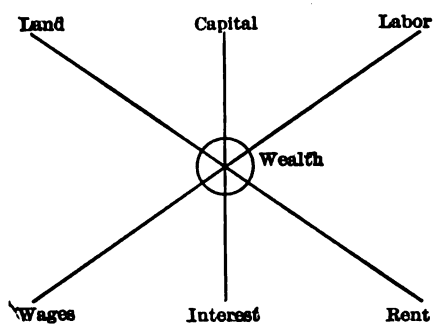
franchises. By labor-product value is meant the value of any handiwork of man; "labor products" include buildings, tools, machinery, land improvements, commodities, etc., and when used productively they are true or economic capital, as distinguished from "capital," used popularly, which often includes both land and labor product.

For the production of wealth (labor products) in any branch of industry three factors and no more are needed: land, capital, and labor. After wealth is produced it is distributed among the three factors of production; labor receiving wages, capital receiving interest, and land receiving rent, according to the following trinitarian diagram:



Given the same quantity and quality of capital and labor, and the wealth output will vary with the productivity of the land utilized. This land difference may be due to geological causes, as with soils or with ore deposits, or to location, as in the case of town sites or public-utility franchises.

2. *The Essential Factors in Fixing Prices.*

The rent of land is essentially a residual product, obtained by subtracting wages and interest from the wealth output; and this is true because both labor and capital must be continually supported and renewed or they perish, while land is self-supporting and self-existent.

In each industry there exists "marginal" land, *i. e.*, land whose product leaves no residuum after wages and interest have been paid. This marginal land varies in quality with commercial activity, but it may be described as the leanest land that has to be worked to satisfy the current requirements of the market. Though supply and demand may name the temporary price of a commodity, its average price is fixed by

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the cost of reproduction on marginal land; hence under normal conditions rent cannot enter into price, for marginal land is rentless. Only in the cases where all available marginal land was monopolized by some owner, who would not allow it to be used at all without the payment of rent, would rent become a factor in determining price.

3. *The Relation of Land Ownership to the Distribution of Wealth.*

Capital and labor are movable, capable of being reproduced, and therefore essentially competitive in nature; while land is immovable, fixed in quantity, and therefore its control implies exclusiveness and monopoly. Thus the land owner has the ultimate control of production, and is also the residual legatee of the output after the claims of labor and capital have been paid. As competition constantly tends to cut both wages and interest down to the lowest living rate, the residual monopolistic factor, or rent, tends to absorb the whole gain proceeding from improved methods of production.

The fact that land ownership carries with it special economic privileges not shared by the laborer or the capitalist was perceived by the ancients; the Hebrews[2] had many laws limiting the right of private property in land, while the land question agitated the Roman Republic[3] for generations.

Though the dual theory of property was long ago expounded by such economic masters as Adam Smith, David Ricardo, John Stewart Mill, and Henry George, it has only recently affected legislation. Even in politically democratic countries the great fortunes still arise rather from the private ownership of valuable land than from great individual activity or ability in production.

4. *The Relation of Land Ownership to Speculation and Development.*

If "speculation" be defined as the holding of property out of use for the purpose of obtaining an enhanced price, it is evident that land lends itself easier to speculation than labor products, because it is imperishable and, being fixed in quantity, tends to advance in price as population and business increase. Speculation in labor products is only profitable where the floating supply can be cornered for the purpose of obtain-

[2] Leviticus, chap. xxvi.

[3] *History of Rome*, by Theodore Mommsen, Book III., chap. 12.

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ing temporarily that unearned profit which land ownership in growing countries tends to give naturally and continually.

The transfer of mining lands to individuals by Federal patent means the loss of public proprietary rights, and thus practically, under present Constitutions, all control of mineral development. It thus becomes a matter of individual interest, or whim, whether the land be worked or whether it be held indefinitely idle. Great tracts of mineral land in the East and timber land in the South and West are held idle by great corporations, who thereby restrict production and increase prices by lowering the quality of available marginal land. Well-known cases are anthracite[4] and coking coal, iron ore,[5] petroleum, bauxite, yellow pine, and Oregon fir.[6] These increased prices make possible the dividends on floods of watered securities and artificially increase the cost of living. In the West the productive mineral districts are generally blanketed by scores of claims, whose owners, though they do nothing themselves, will levy a small fortune as a toll on any would-be developer.

Even in Mexico, where patents are not issued, the law of 1892, which allows mineral land to be held indefinitely by the payment of a moderate rental, has favored the hamstringing of many districts by speculation. As the rental is not heavy enough to penalize idleness, as in Peru, the former Mexican law, which required annual development as a *sine qua non* of land holding, was much better for the working, as contrasted with the gambling, miner.

5. *The Relation of Land Ownership to Political Power.*

The organization of savages and barbarians was based on the family and clan;[7] it was independent of definite land areas. On the contrary, a modern nation is organized on a purely territorial basis; it lives on, whatever the social grouping or location of its inhabitants. It is thus clear that the control of land is the foundation of civilized society. Under the feudal system of mediæval Europe the proprietors of land and the civil governors were the same persons, and land rents and land taxes were synonymous terms.

In feudal England, the land owners were the gentry or gov-

[4] The Coal Trust, by H. Davis. *Everybody's Magazine* (Apr., 1906).
[5] *Report of U. S. Bureau of Corporations*, July, 1911, p. 77.
[6] *Report of U. S. Bureau of Corporations*, Feb., 1911, p. 33.
[7] *Ancient Society*, by Morgan.

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erning class, and they not only gave their own time gratis to the State but supported from their rents the public services of defense and justice, while charity and education were sustained by the church estates. The introduction of allodial tenure during the Crusades and of indirect taxation during the 17th century enabled the gentry to shift the burden of government on to the unprivileged working and commercial classes. In 1700 one fourth of English revenue was still raised by a direct land tax,[8] but by 1820 this proportion had sunk to one-twentieth and the masses were pauperized.[9]

Even now in England, the land-owning gentry recognize some obligation to the State in return for their privileges, for they fill with zeal such underpaid, or unpaid, offices as those of local magistrates, city councilmen, and military officers. The adoption, however, in the United States and in the British colonies of the system of private land ownership, along with political democracy (and officials paid from public taxation raised mostly from wages and interest rather than rent), was an anachronism, which has just begun to attract general attention. Though the suzerainty of the State over its territory is still feebly recognized in the public right of eminent domain, the democratic Teutonic (as contrasted with the imperialistic Roman) view of land ownership as a special privilege, requiring an equivalent return by the recipient in time or money, had almost disappeared from American politics till revived in 1908 by the Conservation movement.

That the American lawmakers of the 18th century did their best to prevent the creation of a landed aristocracy is evidenced by their abolition of entail and primogeniture. The Homestead Land Law of 1867 was likewise an attempt to keep the political power of land ownership diffused, but it was rendered largely ineffective by the policy of wholesale land grants to railroad promoters inaugurated during the Civil War.

The diffusion of control which is possible in the bestowal of agricultural land on individuals in fee is seldom practical for mineral resources. Except for surface deposits like placer gold and clay, the modern mining of minerals involves generally many workmen and much capital. It is desirable that the government retain the fee to the ground, not only for the pur-

[8] *Single Tax Review*, vol. —, p. 30 (Jan., 1913).

[9] *Present Distribution of Wealth*, by C. B. Spahr, p. 10.

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pose of controlling the safety of underground working, but also because the operation of a mine often involves the livelihood of more voters than does one of the great surface estates of Latin America. Private proprietary rights which have little significance when transferred with a small farm may become politically unsafe when transferred with a mining claim, and that such a danger is not merely theoretical is proved by observations of semi-feudal conditions in many American mining communities.

Summary.

On the basis of the above economic and political principles, I will now venture to controvert the assertion of Dr. Raymond that our railroad land grants and mineral land laws were essential to the settlement of the West.

Before 1850, the policy of gratuitous land grants to railroad promoters was foreign to American policy.[10] The land grant to the Illinois Central railroad in the 50s was conditioned upon the payment of 7 per cent. of the gross receipts, and this revenue is still collected regularly by the State of Illinois. It was not until the stress of the Civil War had obscured previous landmarks that the policy of wholesale gratuities to railroad promoters from the Federal domain was begun.

Theoretically, these railroad grants were a gross perversion of legislative power, for what right has any democratic government to bestow the national land heritage upon individuals except for a *quid pro quo?* Practically, the grants failed of their announced military purpose, for trains did not run through from Omaha to California till 1869. The chief result of the land-grant policy was the creation of many overgrown fortunes[11] and the encouragement of the idea that it was more profitable to invest in legislative lobbies than in railroad construction.

Of the Federal domain, the homesteaders have received only 115,000,000 acres as compared with 155,000,000 acres given to railroad promoters.[12] At present, of the total Federal holdings of about 1,500,000,000 acres, less than 500,000,000 acres still remain, and most of this residue is unfit for homesteads. With a sufficient arable area for the support of China's people, we are

[10] Railroad Lan Grants in the United States before 1850; thesis for doctorate published by the State University, Madison, Wis.

[11] *History of American Great Fortunes*, by Gustav Myers.

[12] Industrial Progress of the United States of America, by James Douglas, *Journal of the Royal Society of Arts*, vol. lx., No. 3,079, pp. 2 to 40 (Nov. 24, 1911).

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already feeling the pangs of land hunger, while land speculation has become more profitable than farming, and the price of farm land has more than doubled since 1900.[13]

The Mineral Land Law of 1872 was intended by Senator John Sherman as a means for obtaining Federal revenue, but this object was foiled by Senator Stewart and the Comstock lode speculators, for it finally was born as an "Act to Promote the Development of Mining Resources." This act gives away to the first comer, practically gratis, the Federal mineral domain. Its first effect may have been to hasten the inrush of genuine prospectors, but an experience of 40 years has demonstrated that it is more favorable to the forestaller than to the developer, and at present it is the chief reason for the paralysis of many of our Western mining districts.[14]

Dr. Raymond's question, "Why do we not hear of the leasing of gold, silver, and copper mines, and of agricultural lands?" has already been answered by many foreign countries. Canada profitably leases several of the Cobalt silver mines, while the system of public metal-mine leases is embodied in the Napoleonic code, and prevails throughout Latin Europe and America. Both pasture and timber land are publicly leased on a large scale in Australasia, and timber land alone in Canada and in many European countries.

As to farming land, this is much worked under private lease in nearly every country, while in the United States 37 per cent. of the farms were tilled by tenants in 1910.[13] It makes no difference to the tenant whether the recipient of his rental payment is the State or an individual, and this is shown by the experience of various European cities as farming landlords—for instance, Szeged, Hungary.[15]

Considered from the standpoint of economic equity, the private absorption of individually unearned rent is as repugnant for one class of land as for another; but from the standpoint of political democracy the retention by the nation of the proprietary control becomes the more important for lands requiring the congregation of many voters under one management, as with coal, copper, and iron mines, than for individual operations like small farms.

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[13] U. S. Census Reports, 1900 and 1910.

[14] The Paralysis of U. S. Mining Districts, by E. B. Kirby, *Report of the American Mining Congress* (Oct., 1909).

[15] *The Public*, Chicago, vol. xiii., No. 625, p. 269 (Mar. 25, 1910).

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The further private monopoly of available marginal land, which is now prevalent for the alienated mineral and timber resources of the United States, can constitutionally be prevented at once by the adoption of the public-leasing system, as explained by Dr. Smith. Thus, the artificial enhancement of commodity prices would be hindered, and watered-stock jobbing be discouraged. Under a leasing system, all unused natural resources would remain in the public domain and the *entrepreneur* would not, as now, have to pay some forestaller to first step aside before he could begin development.

Another advantage of the public leasing system should appeal especially to scientific workers. At present the economic work of the Federal Geological Survey inures chiefly to the financial benefit of private land owners, which is inequitable, for it is paid for out of revenue raised by indirect taxation on the consuming masses. With public leasing, the people who pay the bills would also get the benefit of the Survey's work, for increased outputs would mean more royalties for the nation.

G. W. WEPFER, Berkeley, Cal. (communication to the Secretary *):—Having been for the greater part of last year in Chile, Bolivia, and southern Peru, I have devoted my attention and study especially to the mining industry of Bolivia. I have been amazed by the great mineral wealth of that country in gold, silver, tin, copper, antimony, wolfram, etc., throughout the Andean Plateau and the Eastern and Western Cordillera.

When the Spaniards left, they abandoned 10,000 silver mines, all in working order and producing, many in bonanza. There were no difficulties with mines which they had not overcome. The government of Bolivia needed money; the larger part of the nation was much impoverished. The greater the imposts laid upon the mining industry, the less were the results. The mines could not be sold, as there was no capital in the country.

In 1833 the Congress of Bolivia decided upon the leasing system, which is in vogue up to the present time. It is the only system by which Bolivia has obtained for her mining industry a considerable measure of success. The government is expecting more, especially as now three railroads carry ores and metals from Bolivia to the seaports on the Pacific. Besides the direct railroad connections, conveys, consisting of

* Received Mar. 21, 1913.

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from 1,000 to 1,500 llamas, can be seen carrying concentrates of silver, tin, and copper to Chililaya, on Lake Titicaca, for further conveyance to Mollendo, on the Pacific.

The mining claim, or pertenencia, or hectarea, is equal to 1,000 sq. m., or 2.47 acres. Mining is open to all comers, natives and foreigners. The government, to establish a basis for the money value, has an agreement with the Bank of England by which that bank accepts 12.5 bolivianos as equal to £1. This would give to the boliviano a value of 40 cents American currency.

The mine dues, payable semi-annually, amount to bolivianos 2, equal to 80 cents American currency, per 1,000 sq. m. Besides that, the lessee must take out a patent for this mine or mine ground, once for all, which costs bolivianos 10, equal to $4. The lease can be taken out at the office of the Mining Department in the capitals of the respective provinces or at La Paz, the capital of the country. If the mine dues are not paid, the government accepts this as an abandonment of the lease, and the next comer may obtain a lease for the same mine, together with all there is on the mine ground.

The Bolivians are good strong men. The daily wage in most parts of the Andean Plateau is bolivianos 1.50, equal to 60 cents. In the desert places, where provisions have to be brought from some distance, the wages are respectively higher and the mine owner must provide the workmen with habitations, mostly houses built of adobe.

R. W. Raymond, New York, N. Y. (communication to the Secretary *):—I regret that by reason of absence and illness, I did not see Director Smith's criticism of my paper until long after it was published, because it calls for an apology on my part, which should have been made promptly. Regarding his main argument, I must be content, for the present, to acknowledge both its courtesy and its force, without admitting that it has changed my own convictions. Other contributions to the discussion have been received, as I am informed, by the Secretary of the Institute, and await publication. I hope I may be able, at some future time, to review the whole subject once more. Meanwhile, the final passage of Director Smith's contribution requires my immediate attention.

Namely, he points out that I have seriously misrepresented

* Received Apr. 18, 1913.

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Mr. Parker, by saying that he, having "observed the evils of hot competition in the bituminous coal-mines, thinks the government should interfere by ordering those mines to shut down which do not pay." And he quotes from Mr. Parker's printed paper what he "actually said," in the Spokane paper to which I referred.[16]

My reference to Mr. Parker's paper was not a quotation, and therefore expressed only my own summary of his view, as expressed in his paper. Moreover, I frankly confess that my understanding was influenced somewhat by the friendly, colloquial discussion of his paper, when he presented it orally at Spokane. At all events, I cheerfully withdraw now,—as I would have done long ago, if my attention had been called to it,—this or any other statement of mine as to what "Mr. Parker thinks," which seems to him unfair; and I heartily apologize for my presumption and error in making such an unwarrantable deduction from his language.

But Mr. Parker's printed paper, as quoted (apparently with approval) by Director Smith, seems to me to offer some excuse for my crime. His paragraph on this subject (p. 601) begins:

"It is, perhaps, somewhat bold to suggest that the bituminous mines should be put under some sort of government control; but if they are not, I am frankly of the opinion that"—

Here follows a statement of the evil result to be expected, namely, the control of the business by private capital, which will regulate the production according to market requirements. Then follows this significant proposition:

"Under our system of government the Federal authorities have no jurisdiction over mines in the several States, *unless the power given them under the Constitution to regulate commerce between the States could be stretched to apply to coal because of its bearing on interstate traffic.*"

And Mr. Parker then outlines three evils, from which "it does look as if a choice will have to be made," namely: (1) a continuation of the present situation, which will be "bad;" (2) the ultimate control of bituminous coal-mining by a vast "combination of interests," which will be "worse;" and (3) government supervision and regulation (not ownership), which is "problematical." And here occurs the sentence quoted by Director Smith as correctly describing Mr. Parker's conception:

"Under such government control bituminous coal-mining could be regulated through a system of license; and in order that restriction on coal-production may be secured, no license should issue for the opening of a new mine until ample proof is shown that the necessities of the people or of trade require it."

[16] *Trans.*, xl., 596 (1909).

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Mr. Parker regards this system as "problematical," no doubt because, as he has just declared, it would not be possible, under our system of government, unless the Federal power to regulate interstate commerce could be stretched to cover it, because of the bearing of coal on interstate traffic. But he clearly favors it, as a remedy for over-competition in coal-mining, not on the U. S. public domain, but on lands already under private ownership.

Now, what would this system imply? Government licenses to mine coal would of course be granted for limited periods, and subject to Executive revocation upon non-fulfillment of the conditions. Is a licensee to be unconditionally entitled to a renewal of his license, after its term has run out? May not the government then refuse a renewal, on the ground that his product is not required by "the necessities of the people or of trade"? Is there any better proof that his product is not thus "needed" than the fact that his business would not pay? And if he were refused a renewal of license, would he not be "ordered to shut down.?"

But if Mr. Parker's system were introduced to-morrow (the Constitution having been successfully and sufficiently stretched for the purpose), would all the present bituminous coal-mines be licensed to go on? If so, would not the present situation be continued? And if not, would not some mines be obliged to shut down, because they were not "needed," in other words, would not pay?

In short, having begged Mr. Parker's pardon for my unintentional misrepresentation of his meaning, I now say, with the greatest respect for his ability and disinterested patriotism as a government official, that his suggestion of a Federal supervision of private industry on private lands within the States is every whit as bad as my misunderstanding made it out to be. The difference between a bureaucratic decision forbidding a new enterprise and one destroying an existing enterprise, is purely academic, and disappears in practice. Both equally substitute official control for individual initiative and free competition; and both belong to the *ancien régime* of paternal government, from which we dreamed that we had been delivered, after the continuous struggles and progressive victories of the Anglo-Saxon race through many hundred years.

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Fires in Metalliferous Mines.

BY GEORGE J. YOUNG,* RENO, NEV.

(Cleveland Meeting, October, 1912.)

I. General.

THE recurrence of mine-fires in Nevada during the past decade is not only a matter of interest, but also one of considerable concern to engineers and mine-managers. The more important fires may be enumerated as follows:

Forman Shaft fire, Gold Hill, April 21, 1903; shaft-house, machinery, and shaft destroyed; loss estimated at $50,000; cause unknown.

Union Shaft fire, Virginia City, July 14, 1904; shaft-house, machinery, and shaft in part destroyed; loss estimated at $100,000; cause of fire stated as the careless throwing of a match in the rope-house.

Sutro Tunnel fire, Virginia City, Jan. 27, 1909; 700 ft. of tunnel-timbering destroyed, and direct damage of $10,000; cause "probably electric wires."

Belcher Shaft fire, Gold Hill, Aug. 9, 1910; shaft-house destroyed, machinery ruined, and upper part of shaft damaged; damage, $25,000; no cause given.

Belmont Mine fire, Tonopah, Feb. 23, 1911; fire started in winze; 17 men killed.

Giroux Mine fire, Kimberly, Aug. 23, 1911; fire originated in shaft-station from unknown cause; 7 men killed, and $34,521 damage caused.

(NOTE 1.—On June 11, 1912, some time after the present paper was written, a fire broke out in the pump-room on the 2475 station of the Ward shaft, Virginia City. This fire was caused by the short-circuiting and exploding of the starting-switch on No. 5 pump. The oil in the switch was scattered about and set fire to the timbers and lagging of the pump-room. The pump-man in charge notified the surface, and the electric current was shut off. A hose was then turned on the fire, but the dense smoke drove the attendant out. Inspection by the electrician and the shaft-foreman, equipped with oxygen-

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helmets, soon after, showed no flames or excessive heat, but the continued expulsion of smoke indicated that some fire was present. The pump-chamber was then ordered flooded. No estimate of the damage was published and no fatalities occurred.)

It is not the purpose of the present paper to discuss the above examples, but rather to use certain features of them to formulate general plans which might be of use in fire-prevention and fire-fighting.

In the above list three fires are noteworthy in that they originated in wooden shaft-houses and communicated with workings underground, resulting in considerable damage. The risk taken by closing in the mouth of a shaft with wooden buildings has long been recognized, and most of the Western mines have eliminated the shaft-house. In Virginia City, on account of the severe winters, the shaft-house has been retained, but, following the lesson given by these three fires, and the passage of a mining-regulation governing this feature of mine-construction, active steps have been taken to remove the risk. In almost every case where the shaft is used as a working-shaft wooden buildings about the shaft-mouth have been removed.

Mine-fires may be considered in two groups: those which occur in the surface-plant and those which occur in the underground workings. Fires in mine surface-plants do not, as a rule, result in loss of life, and if proper fire-fighting facilities are provided, the fire may be extinguished with but moderate loss. In Nevada considerable progress in the use of fire-proof construction has taken place in recent years. The following steel and semi-fire-proof constructions may be noted: Goldfield Consolidated mill, Goldfield; New Belmont mill, Tonopah; Nevada Consolidated concentrating-mill, McGill; Nevada Reduction Works mill, Dayton; Nevada Hills mill, Fairview; Pittsburgh Silver Peak, Blair.

Steel head-frames: Montana Tonopah, Tonopah; Tonopah of Nevada, Tonopah; Union shaft, Virginia City; Star Pointer shaft, Ely; Merger mines shaft, Goldfield; Giroux mine, Kimberly.

The protection of the surface-plant from fire is recognized as a necessity, and most mining companies, where an extensive plant has been installed, provide facilities of some sort for fire-fighting. In the design of a surface-plant the question of fire-risk should receive considerable attention. The segregation of

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different parts of the plant, the separation of each unit by a sufficient distance to prevent the spread of a fire from one unit to another, and, finally, the use of materials which are either fire-proof or of a slow-burning nature, are the main points which deserve consideration at the start. While it is true that the use of expensive materials and equipment is not warranted in the early stages of a mine's development, it is often the case that after a mine has reached the producing stage the same types and materials of construction are used, and a surface-plant grows until it involves an amount of combustible material that in itself is a risk of some magnitude. The inevitable happens, a fire gets beyond control, and the destruction of the plant follows; this occurs in spite of what were deemed adequate fire-fighting facilities.

That a more general use of fire-proof and semi-fire-proof materials and construction in Western mines is merited goes without saying. That there is progress in this direction no one familiar with the more recent camps of the West can deny. It may not be out of place here to review the different types of building-construction used for surface-plants. They may be enumerated:

1. Timber frame with board siding.
2. Timber frame with corrugated-iron siding and roof.
3. Steel frame with curtain-wall construction in which brick, ferroinclave, and cement plaster or reinforced concrete is used.
4. Reinforced concrete.
5. Brick walls, steel frame, and corrugated-iron roofing.

In Nevada the first two methods of construction are common in the case of most mines and prospects. Of these, the second method is preferable on account of reducing the amount of combustible material. By whitewashing the timber-work both ignition and the spread of a fire may be retarded. The third method of construction is in use where large plants, more or less permanent in their nature, are erected. Examples have been cited above. The fourth method is quite common in coal-mine surface-plants in Westphalia, Germany, but is not often met with in the Western States of America. It merits consideration by the mining engineers of the United States. The use of reinforced concrete is uncommon, but some examples are to be recorded in Nevada. The Midway mill, erected in the early days of Tonopah, was constructed with steel frame

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and corrugated-iron siding, and contained a reinforced-concrete ore-bin. In the Belmont mill and the Goldfield Consolidated mill, reinforced concrete has been used. In Nevada, brick construction, for obvious reasons, is not used for mine-plants, but in the case of electrical installations for power-plant and sub-stations this method of construction is common.

The critical parts of a mine surface-plant are: the blacksmith-shop, boiler- and power-plant, change-quarters, wood-shop, oil-storage, rope-house, and the structures about the shaft-mouth. With order, cleanliness, and a proper segregation of buildings, there is little chance for a fire, and yet carelessness on the part of any individual may be the cause of starting a fire, and consequently protection must be afforded by fire-plugs and hose-reels attached to the water-system. Usually the mine-plant is compactly arranged, and several plugs and hose-reels may serve all purposes. Small fire-extinguishers in each building are also considered necessary.

Underground mine-fires are serious, as in almost every case there is danger that fatalities may result, and the difficulty and danger of fighting such fires is great. In heavily-timbered mines where there is little or no water, the possibility of a small fire spreading throughout a mine is always present. Fortunately, dry mines usually do not have large areas of heavy ground, and heavily-timbered mines are often wet mines.

The causes of fires in metalliferous mines are:

The presence of combustible materials, such as timber, oils, waste.

Carelessness with candles, lamps, and smoking.

Blasting; remnants of smoldering fuse.

Overheated bearings in machinery.

Short-circuiting and overheating of electric wires.

Spontaneous combustion.

Rapid progress has been made in the development of methods of mining "without timber," and many methods involving a small proportion of timber, as compared with that required by the "square set" system, are in use. Even though we may largely eliminate timbered stopes or, by filling, practically eliminate the fire-risk from them, we still have the timbered drifts, stations, shafts, pump-stations, and winzes. We could almost completely eliminate the fire-risk from these by

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the use of steel and masonry, but this is practicable only in a few cases, and the mines of the West will undoubtedly utilize timber for many years to come. Underground, a relatively large amount of combustible material must be contended with. The critical places are the shaft-mouth, the shaft, the stations, the drifts, and the winzes. The critical condition is where the timber is dry and comparatively little water occurs. Where water is encountered the fire-risk is lessened. Systematic elimination of combustible material should be the first thought of the engineer. The proper protection from fire where combustible material must be used is the next consideration.

Carelessness with candles may be eliminated by the introduction of electric lighting, and this method of lighting is common in Western mines of any size. Where candles are used, proper receptacles at stations and in stopes should be provided and their use insisted upon. The use of these will go a long way towards eliminating this cause. The setting of lighted candles upon timbers should be prohibited, and the removal of all candles from working-places when miners are leaving should be required. Lamps should be sparingly used, and where possible these should be filled and trimmed outside of the mine. Oil, and particularly oil required in illumination, should not be stored in the mine. Where oil-lamps are used by the miners, some form of solid illuminant should be used. Smoking can be controlled by mine-regulations, and in a heavily timbered mine, or in and about a wooden surface-plant, it should be prohibited. Where possible, the development of a stable, steady-working crew of men is an important factor in preventing accidents of any kind. Good foremanship is essential to this end.

Blasting as a cause of mine-fires is, no doubt, of minor importance. I know of no authentic case where the flame of a blast has been responsible for setting timbers on fire. In the case of the Homestake fire a piece of smoldering fuse has been given as the cause. However this may be, it is necessary carefully to inspect timbered stopes after blasting. A fire originating from any cause during the interim between blasting and the arrival of the next shift may thus be discovered before it has gained any considerable headway.

Overheated bearings of machinery are an infrequent cause of fire. Underground ventilating- and pumping-machinery are

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the only forms of machinery which would be likely to cause trouble of this nature. Ring oil-bearings on motors and fans have, to a considerable extent, removed this source of danger, but frequent inspection of machinery should be made. Where fans are in use an inspection should be made at least twice a shift by the shift-boss. Ring oil-bearings should be frequently examined and kept filled, and at intervals the oil should be completely removed, and the bearings thoroughly cleaned. Machinery oil should be kept underground in quantities only sufficient for several shifts' use. Machinery, where possible, should be placed in untimbered chambers or, where support of some kind is necessary, masonry, steel, or some fire-proof material should be used. At Virginia City the pump-rooms are heavily timbered and in themselves contain sufficient timber to sustain a considerable fire. The practice is to whitewash these timbers, and to keep such rooms thoroughly clean. While this reduces the fire-risk, still the presence of so much combustible material, and some of it of an oil-soaked nature, must be considered as a risk, and precautions should be taken to minimize it. (See note 1.) Such chambers can be readily provided with hose-plugs and reels. Where machinery of any size is in operation, the presence of attendants is usual, and is an additional safeguard. Oily waste and waste of any kind used in and about underground machinery should have metal containers provided.

Electric wires and apparatus may have been the cause of some fires, but where their installation has been carefully looked after, and they are in the hands of experienced men, it is seldom that a fire can be directly traced to their use. (See note 1.) The proper making of connections, the use of fuses, and automatic circuit-breakers on all apparatus will prevent excessive loads coming upon lines, electric motors, and transformers. At Virginia City transformers are used at several places underground. These are placed in timbered stations, but this practice is open to question. Transformers should be placed in chambers free from timbers, and only material of a fire-proof nature should be permitted in the vicinity. In surface electrical work transformers are placed in fire-proof buildings, and out of contact with wood-work. Where transformers are in use underground, and in the vicinity of combustible material, buckets of

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sand should be placed where they can be used in extinguishing a fire.[1]

Spontaneous combustion is an infrequent occurrence, for the reason that underground conditions are seldom of a nature that would lead to this cause. Oily waste might, under extreme conditions, develop sufficient heat to ignite combustible material in its neighborhood, but it is seldom that this is present. Fires in mines containing heavily sulphureted ores are claimed by some writers to be due to the heat produced by great pressure. While this cause is not of great importance, a careful mine-manager will not overlook it.

II. Fire-Prevention.

The first line of defense in the prevention of fire is a proper set of mine-regulations covering the use of candles, lamps, oils, and other combustibles used underground; the second is the enforcement of these regulations. Without discipline mine-regulations are of little avail. The third line of defense is the practice of a fire-drill at frequent intervals. This practice should include drill in the use of fire-fighting apparatus, the training of a suitable fire-fighting squad, and the accustoming of the men to answer an emergency fire-call, so that a possible panic among the men on the alarm of fire may be avoided. The drill should be segregated into the drill of the fire-fighting squad at least once a month, and the drill of the whole mine-force in answering a fire-call at least once in three months. The fourth line of defense is the use of watchmen, whose business it should be to inspect the mine-workings after a shift has departed or at frequent intervals when the mine is shut down. These men serve to check up careless miners and prevent a small fire from spreading.

Facilities for fighting incipient fires should be provided, and placed at readily accessible points in the vicinity of the places where there is any considerable amount of combustible material. These places would be at shaft-stations, the sill-floors of timbered stopes, timbered winzes, and shaft-collars. Fire-extinguishers of a simple type would be the means provided. Water-pipe line should be laid to large timbered stopes, and hose-connections provided. Hose-reels should be placed at the

[1] The Factor of Safety in Mine Electrical Installations, *Technical Paper No.* 19, *U. S. Bureau of Mines* (1912).

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critical points. The whole system should be standardized so that hose, couplings, etc., could be transferred, and used in any part of the mine. At the surface a reserve supply of hose should be provided. A monthly inspection of fire-fighting appliances should be provided for.

Steel fire-doors, or wooden doors protected by tin sheets, set in concrete bulkheads, should be provided in cross-cuts and drifts at such points as would enable the spread of a fire starting in a stope to be checked. These would only be used in the event of a considerable fire. Such doors must be constructed so as to admit of opening from either side. In important tunnels of any great length and where timber is used several fire-doors might well be placed so as to divide the length into sections. In lieu of fire-doors a length of 100 ft. of the tunnel could be supported by masonry instead of timber, and this would localize a fire. In tunnel-fires fire-doors cannot always be reached, and the conditions may be such that temporary bulkheads cannot be constructed. In such cases the masonry zone might prevent the fire from spreading through the entire length of the tunnel. In railroad tunnel-work, where, from financial conditions, timber must be used instead of masonry, masonry zones might well be used. The recent fire in the Chilcoot tunnel of the Western Pacific railroad is a case in point. A main working-shaft in a mine would admit of fire-localization in the same way. In place of masonry, steel tunnel-sets or shaft-sets could be used. While perhaps general rules for the placing of fire-stops and doors could be formulated, the conditions at each mine would have to be studied and the locations of these safeguards determined in such manner as to best meet those conditions.

O. F. Heizer has informed me that at the Seven Troughs Coalition mine, Seven Troughs, Nev., hinged wooden doors, protected by $\frac{3}{16}$-in. steel sheets, are placed at the second set below the collar of the incline shaft (75° incline). These doors are held by trickers controlled by a lever. Slots are cut so that interference with the hoisting-cable is avoided. Ten sacks of sand are stored close by the shaft. In the event of a fire on the surface the surface-men have instructions to throw the doors and place the sand over them. He informs me that this device worked very well on one occasion when a fire took place in the surface-plant. He also informs me that at the Nevada Hills mine, Fairview, two steel sheets are arranged at the mouth of

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the shaft so that they may be slid over the opening of the compartment by throwing a lever. No special rollers are used, the sheets simply sliding on angle-irons.

To one familiar with conditions in Western mines, it is a matter of surprise that advantage has not been taken of the automatic sprinkler, which is in use for the protection of warehouses, factories, public buildings, and the like. I know of no instances where this system is used underground, and only in a few instances, notably in Montana, have they been used in metallurgical plants. The system as applied to buildings consists of a network of pipe-lines supplied by a pressure-tank. The pipes are so distributed as to admit of a sprinkler-head being placed in the center of each 10-ft. square. The sprinkler-heads are placed close to the ceiling. Each head is provided with an opening which is closed by a non-corrosive button. The button is held in place by a two-piece metal strip. The pieces of the strip are held in place by a soft solder which has a melting-point sufficiently low to be speedily reached by any fire in the vicinity. With the melting of the solder the pieces fall apart, the water-pressure forces the button out and the water spurts out against a spreader which throws the spray over a considerable area. A melting-point of 155° F. is used in the well-known Grinnell automatic sprinkler. The complete installation of an automatic sprinkler-system in a mine, while not necessarily impracticable, would in most cases be unnecessary. As has been mentioned, the critical places are the shafts and shaft-stations where the timbers are dry. Shaft and stations might well be protected by such a system. Timbered pump- or machinery-stations, on account of the value of their contents and their importance to the mine, could also be protected in this way.

The installation of an automatic sprinkler system in a shaft is represented in Fig. 1. Two main points in the design of such a system need consideration: one is the question of water-pressure; the other is the use of protective coating upon the pipe so that scale and rust cannot form and clog the sprinkler-heads. In a deep shaft it is doubtful whether a continuous pipe-line could be employed without the use of extra-heavy pipe and fittings. By placing pressure-tanks at intervals of from 300 to 500 ft., ordinary or "extra-strong" pipe and fittings could be used. The intermediate tanks could be supplied from the pipe-line above by means of valves operated by

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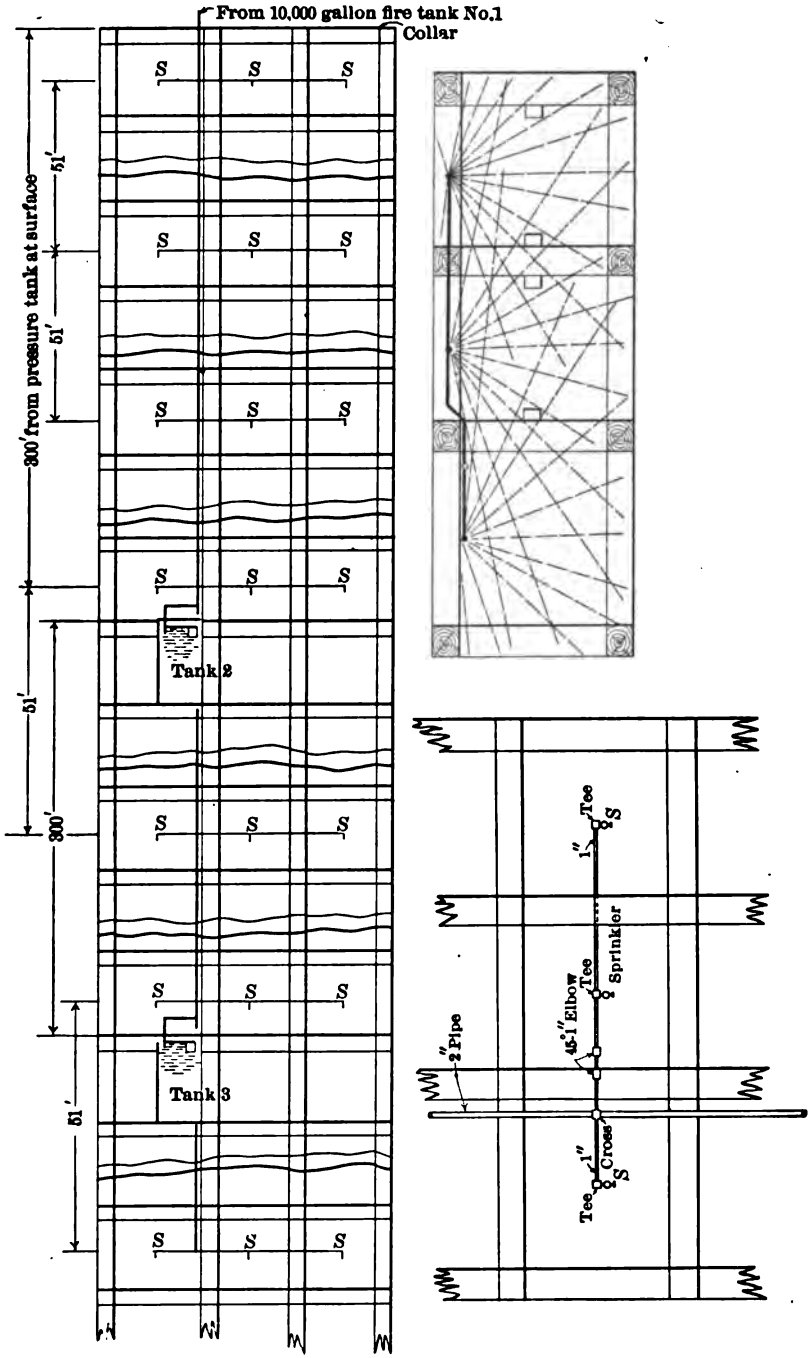

FIG. 1.—AUTOMATIC SPRINKLER INSTALLATION FOR A MINE-SHAFT.

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float. The use of galvanized pipe and fittings would eliminate ordinary troubles of corrosion.

In each compartment of the shaft, at intervals, sprinkler-heads, point downward, would be placed. In order to be out of the way these would be placed back against the lagging, but this position would not prevent their effective operation. A three-compartment shaft would be protected by making the intervals between sets of sprinklers 50 feet.

The cost of such a system is nominal compared to the amount of damage a shaft-fire might cause. I have estimated the cost of pipes, sprinklers, and pressure-tanks (every 300 ft.) for a shaft 1,000 ft. deep to approximate $500, or $0.50 per foot. In Fig. 1 a 2-in. pipe is assumed as sufficient. In the cost of the system should be included a fire-tank on the surface. Shaft-stations would be protected by extensions of the pipe-system along the center-line of the station and the spacing of sprinklers every 10 feet.

The advantage of such a system is its automatic operation. A fire arising in an obscure portion of the shaft would be taken care of and extinguished before it could spread. The shaft, the main exit of a mine, could thus be always protected and maintained in a workable condition. The possibility of saving life in the case of a fire would be greatly enhanced if shafts could without question be kept open.

By extending the sprinkler-system along each drift for 50 or 100 ft. a fire could be prevented from spreading to the shaft.

An automatic system to be satisfactory should be inspected at frequent intervals and such a system, carefully installed, would not require an overall inspection more than once in three months.

A telephone-system is an indispensable part of the equipment of a large mine. Not only is it necessary to install telephones at stations, but important winzes and stopes also should be provided with this appliance. The prompt warning of the men may be the means of preventing loss of life. This could be done by telephone or by the use of flash-signals where incandescent lamps are used. In Virginia City an "all out of the mine" signal is in use, and the electric lamps by a given number of flashes convey this signal.

Fire-fighting helmets are now considered necessary at all large mines, and the training of men in their use should be a part of the fire-drill. Oxygen-helmets also should be provided

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for possible use in rescue-work as well as for the attack of smoldering fires which would produce such an amount of smoke as to prevent near approach. So well has the subject of the construction and use of the oxygen-helmet been discussed that further comment is unnecessary here.[2]

All underground air-lines should be provided with connections at stations, winzes, and stopes. These connections should be maintained in working-order at all times. In the Belmont fire a connection on the air-line at the 1100 station would have been the means of saving life. Where a water-pipe line is not in use in a working-shaft, connections should be made at the surface, so that the air-line could be used for purposes of bringing water to a fire. It is preferable to have two lines of pipe in a shaft, one for water and one for compressed air.

III. Fighting Mine-Fires.

The first essential in fighting a mine-fire is to get all of the men out of the mine. When this has been accomplished, a plan of action can be decided upon and carried out under competent direction. The usual steps may be stated as: bulkheading, with the object of localizing the fire; laying of hose and the bringing of a water-supply to the scene of the fire.

The fire-fighting squad necessarily works upon the incoming-air side of a fire unless equipped with smoke- or oxygen-helmets. By carrying air-lines along with hose-lines, fire-fighting squads have been enabled to approach close enough to a fire from the "lee side" to do effective work. It requires courage and daring on the part of the fire-squad to perform work of this kind, but instances are not uncommon where stubborn fights of this kind have been made.

Where it is impossible to approach the fire close enough to fight it with water, two methods may be used: one is the bulkheading of the fire on both sides and the closing of all winzes leading from the fire-zone; the other is to seal the mine and fill the workings with a gas which will prevent combustion. In the former method the fire is left to smother out, and this may take considerable time. The availability of the oxygen-helmet renders it possible to construct bulkheads where it would have been practically impossible without the use of this appliance.

[2] The Use and Care of Mine-Rescue Breathing Apparatus, *Miners' Circular No. 4, U. S. Bureau of Mines* (1911).

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In fighting a fire by the second method, steam, sulphur dioxide, and carbon dioxide have been proposed as gases suitable for the purpose. Steam is the agent most used. If a supply of sulphur could be speedily obtained, it might be possible to use this reagent in temporary burners arranged so as to discharge the gases into the intake air-ways. Snelling[3] has given details of the method. Carbon dioxide is difficult to generate in sufficient quantity, and its use is almost out of the question save as it is generated by the fire itself. Where it is impossible to bulkhead a fire, and steam or other agent is out of the question, the flooding of the mine is the next expedient. If it is impracticable to flood the mine, the turning of water down the shafts after sealing all openings to the mine is then in order.

The great danger in all mine-fires is the rapid filling up of the workings with smoke and poisonous gases (CO). To one who is familiar with fires this is the most striking thing. The comparatively restricted workings of a mine fill up in a very short time on account of the air-currents, which, while normally moving sluggishly, under the increased temperature rapidly acquire velocity. The presence of fire-doors at intervals may be the means of preventing workings from being completely flooded with irrespirable gases. If it were possible to stop all air-currents in a mine by doors suitably placed, the fire and the gas could be prevented from spreading outside of a restricted zone.

As in the case of surface-fires, "being prepared" is the key to the situation. Mine-superintendents should carefully consider the possibilities of a fire, and make every preparation beforehand, even to laying out a method of procedure for fires occurring in different parts of the mine. The drafting of a plan of action, the provision of apparatus and means for carrying out this plan, as well as the drilling and training of the men who are to carry the plan into execution, will go a long way towards preventing confusion, delay, and loss of control at the fire-signal. The provision for marking the passages leading to shafts and exits should be carried out in all large mines. A careful study of the ventilating-currents in a mine and a consideration of the effect of a fire in changing their direction also form necessary parts of any fire-fighting plan. Ventilating-

[3] *Trans.*, xxxix., 550 (1908).

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plans of the mine under varying conditions should be prepared, and their study made a part of the drill by the fire-fighting squad. The effect of turning a stream of water down a shaft upon the air-currents deserves mention. In most cases a stream of water turned down an upcast shaft has the effect of reversing the air-current. In the Giroux fire the upcast shaft which was on fire was provided with a water-pipe, pierced with holes, close to the collar of the shaft. An attendant at the surface (without authorization) turned the water into this pipe, and thus reversed the direction of the air, causing the shaft to act as a down-cast. The Alpha shaft, which was the down-cast, became an upcast, and the miners escaping by this shaft were killed by the gases.

IV. Legal Regulations.

An examination of the mining-laws of the Western States indicates no general tendency to comprehensively cover the subject of mine-fires. As an example, the regulations provided by the State Mine Inspector Law of Nevada may be cited. These regulations are grouped as follows:

Regulations Relating to Egress from Mines.

"Section 19.—All shafts shall be equipped with ladders, and shafts more than 200 feet in depth, inclined more than 45 degrees from the horizontal, equipped with hoisting machinery, shall be divided into at least two compartments; one compartment to be divided off and set aside for a ladderway. The ladders shall be sufficiently strong for the purpose demanded, and landings shall be constructed not more than 30 feet apart; said landings to be closely covered, except an opening large enough to permit the passage of a man. A landing shall be constructed in manway at all working levels."

"Section 20.—In every mine within this State, if more than 200 feet in depth, where a single shaft affords the only means of egress to persons employed underground, and the ladderway compartment is covered by a non-fireproof building, it shall be the duty of the operator of said mine to cause said ladderway to be securely bulkheaded, or a trap door placed over same at a point at least 25 feet below the collar of the shaft, and if a trap door is used, it must be kept closed, or so arranged that it can be closed from a point outside of the building by the releasing of a rope, and below this bulkhead or trap door, if the shaft is situated on a side hill, a drift shall be driven to the surface, and if the shaft containing said ladderway may be otherwise situated, this drift shall be driven on the level to a safe distance, but in no case less than 30 feet beyond the walls of the building covering the main shaft, and from such a point a raise shall be made to the surface. The said raise shall be equipped with a ladderway, and it, together with the drift connecting with the main shaft, shall be kept in good repair, and shall afford an easy exit in the event of fire."

"Section 21.—Whenever the exit or outlet from a mine is not in a direct or continuous course, signboards, plainly marked, showing the direction to be taken, must be placed at each departure from the continuous course."

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"Section 28.—It shall be the duty of every operator to provide every tunnel or adit level, the mouth of which is covered by a house or building of any kind, with a door near the mouth of the same, that can be closed from the outside of the building by a pull wire or cable in the event of fire; inside of door a raise shall be run to connect with surface, thus affording a means of exit in the case of fire."

Regulations Relating to Structures over Shaft-Mouths.

"Section 27.—It shall be unlawful for the operator of any mine within the State to erect any structure over the shaft of any mine, except head-frames necessary for hoisting from said shaft or outlet, and the hatch or door necessary for closing such shaft or outlet; *provided, however*, it shall be lawful to erect a house of non-inflammable and fire-proof material over such shaft or adit to protect the men working at such point. In the case of existing houses covering mouths of shafts or adits, it shall be the duty of the superintendent of the mine to cause the immediate removal of all inflammable material stored therein, and it shall be the further duty of such superintendent to prohibit the storage of any inflammable material 30 feet from the exterior walls of any housing hereinafter built."

Regulations Relating to Inflammable Materials Used Underground and on Surface.

"Section 16.—All timber removed shall, as soon as practicable, be taken from the mine, and shall not be piled up and permitted to decay underground."

"Section 22.—Use of gasoline underground is forbidden."

Regulations Relating to Fire-Fighting and Control of Mine-Fires.

"Section 41.—At every mine in this State, employing forty or more men underground, there shall be kept on hand at all times, in good working condition, at least two smoke helmets of a design to be approved by the State Mining Inspector, and which helmets shall at all times be subject to his inspection. For every additional fifty men so employed an additional smoke helmet shall be provided."

These regulations are, at best, only fragmentary, but they indicate an effort to provide certain things that a mine-operator must do. Most mine-operators are willing to conform to any reasonable regulations, but until a more or less complete code is drawn up and incorporated in our State mining-laws, we must expect to find them somewhat backward in taking the initiative, except in those cases where either bitter experience or broad training has stimulated operators to take the subject up in detail.

V. Proposed Regulations.

With the object of inviting discussion, and of giving point to the generalizations in the foregoing, I have written the following regulations. The difficulty of providing for every

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emergency that may arise in a mine-fire, and of meeting the miscellaneous conditions that are present in Western metal-mines, is apparent, and in preparing these regulations I have endeavored to maintain a conservative rather than an extreme position.

1. *Egress from Mines.*

For mines deeper than 200 ft., and employing ten or more men underground, operators must observe the following:

1. Two outlets must be maintained in good condition at all times during the operation of the mine.

2. Where two or more shafts are in use, two of such shafts must be provided with ladder-ways in separate compartments, and such ladder-ways maintained in good condition during the operation of the mine.

3. Where ladder-way shafts are inclined at a greater angle than 45°, landings closely boarded, and with openings just sufficient to allow the passage of a man, must be provided at intervals not greater than 30 feet.

4. In mines operated through three or more working-shafts, where each shaft is provided with hoisting-machinery, a ladder-way may be provided in but one shaft.

5. Between levels at least one winze must be provided with a ladder-way, and the same maintained in proper condition where the level is in use for extraction of ore, or ventilation.

6. On each level the direction to shafts and winzes, used as exits in cases of emergency, must be clearly indicated by sign-boards.

2. *Mine Surface-Structures.*

1. None but fire-proof buildings are to be permitted over shaft or tunnel-mouth.

2. Where shaft-mouths are open, wooden head-frames and bins may be permitted.

3. Non-fire-proof buildings must not be placed nearer than 50 ft. to any shaft-mouth or tunnel-entrance.

4. Mine-plant buildings of non-fire-proof construction must be separated from each other by a space not less than 30 ft. wide.

5. In a mine surface-plant where non-fire-proof construction is employed, hoisting-machinery, boiler-plant, blacksmith- and machine-shop, timber-framing shop, change-house, and storage

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of supplies must be placed in separate buildings separated by a fire-space not less than 30 ft. wide.

3. *Inflammable Materials on Surface and Underground.*

1. Where fire-proof shaft- and tunnel-houses are in use, timber or other inflammable material in excess of sufficient for one shift's use must not be stored in such buildings.

2. Timber-yard or timber-storage sheds must be placed not less than 75 ft. from any mine-building.

3. Lubricating-oils and inflammable fluids must be stored in fire-proof buildings separated by a distance of not less than 75 ft. from other mine-buildings.

4. Metal containers must be provided for the storage of all waste used in wiping and cleaning machinery. They are to be provided for each machinery-room in the surface-plant, and for each group of machines underground.

5. Lubricating- and illuminating-oils in excess of a 24-hr. supply must not be stored underground or in surface-buildings other than the oil-storage building.

6. Gasoline and substances of like nature are not to be used underground.

7. Timber in excess of a 24-hr. supply is not to be stored underground.

8. Powder-drifts which are used for the temporary storage and handling of powder underground, must be cleared of all paper, empty boxes and rubbish, at least once each 24 hours.

9. Where timber is framed at one or more points in a mine, these places must be cleared of chips and rubbish, at least once each 24 hours.

4. *Surface and Underground Regulations.*

1. Where mine surface-structures are of non-fire-proof construction a sufficient water-supply must be provided for fire-fighting purposes. One fire-plug and hose-reel must be placed not less than 25 ft. from the shaft-mouth, and such other fire-plugs placed as, in the judgment of the State mine-inspector and mine-superintendent, shall be sufficient to quench any ordinary fire.

2. At shaft-house, in timber-framing building, power-plant, change-room, and such other parts of the surface-plant as the mine-inspector may decide, at least two fire-extinguishers of

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an approved type must be placed and maintained in proper working-order. They must be placed in a conspicuous and convenient place.

3. Smoking shall be prohibited in non-fire-proof surface-plants.

4. Where more than a nominal number of lamps using illuminating-oils are in use a separate lamp-house shall be provided and all filling, cleaning, and trimming done in this building.

5. Where water-pipes are not installed in shaft, connections must be provided with the surface water-system so that the compressed-air pipes may be used to bring water underground in the case of a fire.

6. Connections must be made with air-pipes at each station, whether the same is in active use or not. These connections must be such that air can be turned into the station.

7. Where candle illumination is used underground metal sconces must be provided at all timbered stations and stopes and miners required to use same.

8. The placing of lighted candles on timbers without proper protection is prohibited.

9. Smoking is prohibited in timbered mines.

10. At all timbered stations which are in active use and where water-supply and hose-lines are not provided, at least two fire-extinguishers of approved type shall be placed and maintained in working condition.

11. Where electric illumination or power is used underground or in surface-plant, line-installation and protection must be in accordance with the Electric Code of the National Board of Fire Underwriters.

12. Transformers, where used underground, must be placed in fire-proof chambers and, at each bank of transformers, sand-boxes must be placed and a supply of sand maintained.

13. In heavily-timbered stopes fire-inspection must be provided after each shift leaves and before the new shift comes on.

14. Air-tight doors shall be constructed in each level, where practicable, between upcast and down-cast shafts. Such doors should be preferably of fire-proof construction.

5. *Regulations for Fire-Fighting and Control of Mine-Fires.*

1. The mine-foreman, under the direction of the mine-superintendent, shall have charge of all fire-fighting operations.

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2. The mine-foreman shall designate certain assistants to constitute a fire-fighting squad, and assign duties to such assistants. The names of such assistants shall be posted at shaft-mouths as the fire-fighting squad.

3. The mine-foreman shall conduct a general fire-drill at least once each three months and at that time examine, test, and report to the superintendent the condition of all fire-fighting apparatus, exits, and ladder-ways. This report shall be in writing, and shall receive the signature of the State mine-inspector on his next inspection-visit after the fire-drill.

4. In mines employing 50 men, at least two smoke-helmets of an approved type shall be maintained in proper working-con dition at all times. For each 30 men in addition an additional smoke-helmet shall be installed and maintained in working-condition. The foreman shall instruct the fire-fighting squad in their use.

5. In mines employing 200 or more men underground, at least two oxygen-helmets shall be maintained in proper working-condition. The foreman shall designate certain men for instruction and practice in the use of such helmets.

5a. In mines employing 200 or more men underground, a "pulmoter" in proper working-condition must be provided. The foreman shall instruct the fire-fighting squad in its use.

6. A general signal to indicate a surface-fire and one for an underground fire shall be designated.

7. Where practicable, an "all out of the mine" signal shall be designated and used in emergency. This signal must be used only by persons to be designated by the superintendent.

8. Detailed instructions to surface and underground men as to what to do in the case of fire are to be posted at shaft-mouth and underground stations.

VI. Conclusion.

This paper is written in the hope that some discussion will result. The use of fire-proof construction, the comparative cost of such construction, methods of preventing the spread of, and of fighting, mine-fires, and mine-fire regulations are subjects upon which many of the members of the Institute can no doubt contribute much of importance.

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The Ultimate Source of Metals.

BY BLAMEY STEVENS, TEMASCALTEPEC, MEXICO.

(Cleveland Meeting, October, 1912.)

IT is now generally agreed that most metals have been brought to the surface of the earth by volcanic agencies. The question as to how these metals came from the volcanic matrix to the mineral deposit has been often discussed.

It might be useful to give more serious consideration to the ultimate origin of the volcanic material. It is possible that some light might thus be shed on the persistent associations of some metals with certain particular kinds of rocks. For example, the very general association of gold with acid types of igneous rocks, of tin with granites or quartz-felsites, of certain types of deep-seated copper-deposits with magnesian eruptives, and so on.

One interesting question is as to whether the metals brought near the surface by volcanic agencies appear here for the first time, or whether they came from the central core of the earth. The interior of the earth is, without doubt, much more highly metallic than the crust, and so the core is often thought to be the origin of many of the metals which are uncovered by man.

The Differentiation Theory.—Although a great many theories bearing upon these matters have been discussed,[1] the only one now considered as worthy of much consideration among American geologists is the "Theory of Differentiation of Igneous Magmas."[2] According to this theory, lakes of molten magma are supposed to exist very far down beneath the earth's surface. In these lakes a process of differentiation is supposed to be going on whereby the liquid magma is split up into two or more liquid phases. Each of these phases is supposed to take with it the metallic constituents for which it has the greatest affinity.

[1] Dutton, Hawaiian Volcanoes, *Fourth Annual Report, U. S. Geological Survey*, pp. 75 to 219 (1882–83).

[2] Iddings, *Igneous Rocks*, chap. vii. (1909).

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This theory is an endeavor, in the most simple way, to account for the emission of a varying and widely-different series of igneous rocks from nearly, if not quite, the same vent. The older idea of separate reservoirs was no longer feasible when there appeared to be so many necessary—for instance—seven at Tonopah and nearly twice as many at Goldfield. And thus the idea of the gradual splitting up of one magma was reached.

Like the old simple theory of the sun and stars moving around the earth, the differentiation theory cannot be definitely disproved, but it involves such a reversal of physical conditions, as we know them, from our experience on the earth's surface, that the probability of truth of the theory is reduced to a very small fraction.

The physical conditions which, according to the differentiation theory, are necessary, are that a liquid silicate magma shall split up into two or more liquid silicate magmas. On the surface of the earth we know of no case where this is so. If pressure and temperature could be considered to alter these conditions for silicates in general, it is very unlikely that some extreme type of silicate would not exhibit the same phenomenon on the surface of the earth.

Moreover, the same generalization applies to other than liquid silicates; for example, liquid sulphides, liquid arsenides, and also, where we have had a lot of experience, with solutions in water.

It is true that variations of physical conditions, such as heat, gravity, or electric state, in different parts of a solution, may produce corresponding variations in its composition. These effects must, however, be so small as to be negligible in magma reservoirs. If such reservoirs were of large dimensions, convection-currents, due to loss or gain of heat, would entirely nullify any such differentiation effect.

Fractional crystallization is not seriously advanced as a general explanation of the emission of lavas of varying composition from the same vent, and there is no reason for discussing this process at length. To illustrate what is meant by fractional crystallization, we might imagine a garnet to be melted up and then slowly cooled. It would not again crystallize out as a garnet, but as two or more silicates, or as silicates and a residual magma, or glass. As the silicates are formed, the

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residual magma changes in composition, and with a fractional crystallization theory would be considered to be extruded at periods during such a history. Fractional crystallizations of aqueous liquids carrying silica and other materials have been formed in the end-stages of solidification on comparatively large scales, but these aqueous crystallization phenomena cannot be advanced as any general explanation of the varying composition of successive volcanic outpourings. A proof of the general mixing effect which obtains in magmas, except during the last phases of total solidification, lies in the fact that the large silicate crystals (phenocrysts) which are formed, remain more or less evenly distributed throughout the magma.

Fractional crystallization as a means of segregation has been given as an explanation of certain peripheral deposits of pyrrhotite and nickel-ore. Even this peripheral pyrrhotite-nickel type of deposit, which is described in great detail by Vogt,[3] may now be equally well explained on the more modern and more rational emanation theory; that is to say, they are more likely to be interstitial and replacement depositions made in the solidified periphery by emanations from the still-solidifying central portion of the intrusion.

Some titaniferous iron-ores have been cited as cases of either differentiation or fractional crystallization, and it is quite possible that this may be the case, but titaniferous iron-ore is not a silicate, and proof of its differentiation from silicates would be no proof of the possible differentiation of silicates.

Miscibility of Liquids.—The tendency of liquids to mix is not due to affinity but rather to cohesion. Liquids possess the quality of cohesion in common with solids, and the property of continuous rearrangement of molecules in common with gases. Any liquid exhibits the property of cohesion among its own constituent molecules. When two liquids are placed together the cohesion may be greater among unlike molecules than among like molecules. In this case the liquids are miscible.

It is well known that in the smelting-furnace a liquid metallic lead-bullion differentiates itself from the liquid silicate slag; in other words, the liquids are immiscible.

[3] *Zeitschrift für praktische Geologie*, vol. i., pp. 4 to 11, 125 to 143, 257 to 284 (Jan., Apr., July, 1893).

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No one has ever known, however, of a liquid silicate splitting into two liquid silicate constituents; in other words, liquid silicates are miscible in all proportions.

High temperatures tend towards miscibility; that is to say, away from differentiation. If liquids were apt to contract on differentiation, physical conditions might be so changed at great depths as to render the differentiation of silicates possible. As a matter of fact, when two liquids are mixed there is no known increase of volume large enough to be considered as even a small factor tending towards differentiation.

Mallet's Theory.—In 1875 R. Mallet[4] tried to show that igneous fusion is due to horizontal compressive crushing in the earth's crust. He calculated the energy necessary to crush rock into an impalpable powder, and showed that enough heat could thus be generated in the world to account for the fusion of all the lavas and igneous intrusions which are being made.

It was subsequently pointed out, however, that the heat so formed could not be concentrated locally so as to produce a sufficient elevation of temperature for the fusion of rocks.

Stevens's Extension.—I have shown[5] that the extension of moderately inclined fissures to considerable depths implies that much superior stresses are required to break rocks at great depths than near the surface. Consequently, a great deal more heat may be developed in the crushing of a deep-seated rock-mass than was formerly contemplated. Moreover, this crushing effect can be repeated over and over again on the same material, producing each time an additional amount of heat. The squeezing of a mass of sand might illustrate this to the average mind. Work is certainly done on the sand and turned into heat, but no particle of sand is necessarily broken or worn. By this crushing effect, under great pressure, the cohesion of a rock-mass is not necessarily reduced. The rocks at great depth are not, in fact, crushed into powder, but forced to flow like plastic material. Pebbles in crushed conglomerates are often considerably flattened, and have even been known to be reduced to the thickness of a leaf.

This form of flowage may be brought about by the yielding

[4] *Philosophical Transactions of the Royal Society of London*, vol. clxiii., pp. 147-227 (1873).

[5] *Trans.* xl., 475 (1909).

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of the material beyond the elastic limit, or by the agency of aqueous or other solutions in the pores of the rocks. Such solutions tend to dissolve material from the crystal-faces which happen to be nearly at right angles to the direction of greatest stress, and to deposit it on the faces which are more nearly at right angles to the direction or directions of least stress. In order to estimate the energy transformed to heat, it does not, however, matter how the distortion comes about if we can ascertain the amount of stress and distortion involved.

In order to define our ideas let us represent the mean vertical pressure by DH, where H is the depth at which the distortion is taking place and D the mean density of the overlying rock. Let the greatest horizontal pressure be DHI, where I will be the ratio between the greatest and the least stresses. The stress difference which caused the distortion is then $DH\,(I-1)$.

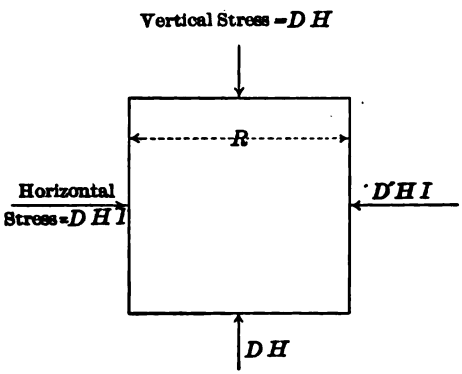

FIG. 1.—DIAGRAM OF STRESSES.

Suppose this pressure difference to compress a unit cube of the rock, Fig. 1, so as to make its least breadth R, instead of unity. As the block must expand inversely as its thickness, the area on which the stress difference now acts is $\frac{1.}{R}$

The increment of energy turned into heat for a decrease $-dR$ of the least dimension will then be $\frac{-DH(I-1)\,dR}{R}$, so that the total energy of distorting the original unit cube of rock is $-DH\,(I-1)\,\log_e R$.

If J be the mechanical equivalent of heat, the amount of

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heat formed per unit weight of rock will be $\frac{-H\,(I-1)\,\log_e R.}{J}$.

From this formula I have prepared Table I., which is intended to appeal to the judgment of geologists on the general probability of truth of the theory.

TABLE I.—*British Thermal Units Evolved by Various Stress Differences and Deformations* $= \frac{H\,(I-1)\,\text{Log}_e\,R.}{J}$

Stress Difference in Feet of Rock Head = H (I-1.)	Deformation Ratios = R.						
	½	⅔	1/10	1/20	1/50	1/100	1/1000
1,000	1	2	3	4	5	6	9
2,000	2	4	6	8	10	12	18
5,000	4	10	15	19	25	30	44
10,000	9	21	30	39	50	59	89
20,000	18	41	59	77	101	118	178
50,000	45	103	148	193	251	296	444
100,000	89	206	296	385	503	592	888
200,000	178	411	592	770	1,006	1,184	1,776
500,000	445	1,028	1,480	1,925	2,514	2,959	4,439

Yield-Point of Rocks.—Much of the distortion of rocks may be due to metasomasis or the re-formation of crystals, but microscopic examination of nearly all distorted rocks shows that the crystals are permanently strained and therefore the yielding-stress is a criterion.

The phenomena of yield in metals has been studied by Prof. J. A. Ewing and W. Rosenhain,[6] who find that it consists of a number of minute shears which occur mostly along the cleavage- or gliding-planes of the crystals.

So far as I know, the effect of other than simple stresses on yielding has never been experimentally studied. This is owing, no doubt, to the difficulty involved in simultaneously measuring stresses in more than one direction on an experimental specimen.

Darwin[7] assumed that the stress difference necessary to yielding was constant, but, as first pointed out by Hodgkinson,[8] this

[6] The Crystalline Structure of Metals, *Philosophical Transactions of the Royal Society of London*, A, vol. CCCV., pp. 279 to 301 (1900).

[7] Thomson and Tait, *Treatise on Natural Philosophy*, p. 423 (1893).

[8] Rankine's *Applied Mechanics*, 4th edition, p. 303 (1868).

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is contrary to the deduction from crushing-tests, where the rupture is usually a shear.

Frictional Stability.—In a former paper,[9] I showed that as soon as these yielding cracks or shearing-planes are formed, the rock-mass becomes subject to the laws of frictional stability of a granular mass, as deduced by Rankine; and in general the ratio of greatest to least stress becomes $I = \frac{1 + \sin \phi}{1 - \sin \phi}$ where ϕ is the angle of repose of the granular material.

It will be noticed that I is determined only by the frictional angle or by the coefficient of friction. Friction is one of the most constant physical quantities we know of. The smallest quantity of solid matter we can weigh has about the same angle or coefficient of friction as the largest quantity we can weigh. We are therefore justified in assuming that some degree of constancy will be maintained up to pressures which destroy the solid properties of the material.

Determined experimentally,[10] ϕ is the angle with the horizontal to which a plane may be tilted before any given object will slide down on it.

For conditions at comparatively small depth between freshly broken rock surfaces ϕ is about 42°, and therefore for simple rock-pressures $I = 5$, approximately, or, if the gravitational water-pressure is added and the greatest stress is assumed to be horizontal, $I = 3.5$. According to the evidence afforded by intrusion[11] of magmas, I is always reduced where there is evidence of plastic flow. It is impossible to say what the exact value may be, but it seems reasonable to put it at about 2.0.

Table II. gives the values of the stress difference as rock head for several values of H and I. This value of the stress difference is to be used to find the heat formed, from Table I., or the corresponding temperature increment, from Table III.

Neither Table I. nor Table II. is based on any assumptions; they are practically absolute. I have purposely constructed these tables in such a way that any person interested may place what value he pleases on H, I, and R and judge if the amount of heat formed, according to the tables, is sufficient to produce fusion.

[9] *Trans.*, xl., 475 (1909).

[10] *Applied Mechanics*, 4th edition, p. 212 (1868); also *Philosophical Transactions of the Royal Society of London*, vol. cxlvii., pp. 9 to 28 (1857).

[11] *Trans.*, xli., 650 (1910).

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TABLE II.— *Values of Stress Differences (I–1) H for Various Values of H and I.*

Vertical Depth or Head, *H.* Feet.	Stress-Ratio, *I* (for Simple Rock-Pressures).						
	1.1.	1.2.	1.5.	2.0.	2.5.	3.0.	5.0.
1,000	100	200	500	1,000	1,500	2,000	4,000
2,000	200	400	1,000	2,000	3,000	4,000	8,000
5,000	500	1,000	2,500	5,000	7,500	10,000	20,000
10,000	1,000	2,000	5,000	10,000	15,000	20,000	40,000
20,000	2,000	4,000	10,000	20,000	30,000	40,000	80,000
50,000	5,000	10,000	25,000	50,000	75,000	100,000	200,000
100,000	10,000	20,000	50,000	100,000	150,000	200,000	400,000
200,000	20,000	40,000	100,000	200,000	300,000	400,000	800,000
500,000	50,000	100,000	250,000	500,000	750,000	1,000,000	2,000,000

Elevation of Temperature.—The terms heat and temperature must not be confused. For example, to fuse 1 lb. of ice from and at 32° F. requires 143.5 B.t.u. of heat, but it does not require any change of temperature. The figure 143.5 is called the latent heat of fusion of ice in Fahrenheit units. An additional amount of heat is required if the material has to be raised to fusion-temperature before being fused. For ice it takes 0.504 B.t.u. for every degree Fahrenheit which 1 lb. of the material is raised before being fused. The figure 0.504 is called the specific heat of ice in Fahrenheit units.

As it is known that the earth is solid practically to the center, it follows that the fusion-point of rocks must rise with the depth so as not to be overtaken by the increment of temperature with depth which obtains in portions of the earth where metamorphism and vulcanism are not proceeding. If this were not the case, all rock magma below 150,000 ft. would be in a state of fusion.

If the deformation be measured on solid rock which has not been melted, the heat-values given in Table I. are the heats necessary to raise the temperature from that which is normal, at the depth where fusion takes place, to the temperature of fusion. After the fusion-point is reached and a small quantity of rock is reduced to the liquid state, the deformation takes place preferably in the semi-molten or pasty material, which is thereby superheated beyond the fusion-point. The melted rock immediately gives up its excess of heat to the surrounding rocks, and supplies the heat for fusing more rocks. Thus,

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if the stress-ratio (R), determined by measurement of schistose deformations around a batholith, is $R = \frac{1}{10}$, and if the rock head (H) is estimated at 100,000 ft., and the stress-ratio (I) at 2, the stress difference is seen from Table II. to be 100,000 ft. rock head, and the heat necessary to raise the temperature from normal (for the depth considered) to fusion-point (for the depth considered) is 296 B.t.u. This at a specific heat of 0.2 gives $\frac{296}{0.2} = 1{,}480°$ F. difference of temperature. If the fusion-temperature can be reached by the heat generated, it is clear from what has been said above that the latent heat of fusion will be forthcoming. I have therefore added Table III. showing the temperature increments corresponding to the various deformation-ratios and stress differences which bring the rock to fusion-temperature.

TABLE III.—*Rise in Temperature, in Degrees Fahrenheit, Induced in Rock by Various Stress Differences and Deformations (Specific Heat Assumed = 0.2).*

Stress Differences in Feet of Rock Head.	Deformation-Ratios = R.						
	$\frac{1}{2}$	$\frac{1}{4}$	$\frac{1}{6}$	$\frac{1}{8}$	$\frac{1}{10}$	$\frac{1}{100}$	$\frac{1}{1000}$
1,000	4	10	15	19	25	30	44
2,000	9	21	30	39	50	59	89
5,000	22	51	74	96	126	148	222
10,000	45	103	148	193	251	296	444
20,000	89	206	296	385	503	592	888
50,000	223	514	740	963	1,257	1.480	2,220
100,000	445	1,028	1,480	1,925	2,514	2,959	4,439
200,000	890	2,056	2,960	3,850	5,028	5,918	8,878
500,000	2,225	5,140	7,400	9,625	12,570	14,795	22,195

Heat Necessary for Fusion.—The heat necessary for the fusion of rocks must vary with the depth, and also with the kind of rock. The temperature of fusion is necessarily higher in depth, but the latent heat of fusion will be less, and the natural temperature of the undisturbed rock greater.

At the surface, the normal fusion-point of igneous rocks is from 1,000° C. to 1,200° C., or from 1,800° F. to 2,200° F., and the specific heat from 0.2 to 0.3.

Others may place what limitations they think necessary on the various factors, but they will doubtless find that there is usually sufficient heat generated to account for igneous fusion.

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Evidence in Volcanic History.—It has been found that, historically, a volcano generally begins with the ejection of a lava which is neither very base nor very acid in composition. It tends, in the course of time, either towards a very acid extreme or towards a very base extreme of lava composition.[12]

On the surface of the earth the basalts are the most fusible rocks, those which are either more basic or more acid being less fusible. With depth, however, there are two very important modifying conditions.

First, the solution of silica in aqueous material tends to lower the melting-point of the more acid rocks. The contraction[13] which takes place when this occurs accentuates this lowering of the fusion-point when high-pressure conditions are superimposed. Second, the greater expansion of the basalts at the instant of fusion makes them less easily fusible when under high pressure than rocks of mean acidity.

It is therefore probable that the first rocks which become sufficiently liquid to flow are those of mean composition, such as the andesites, and that only as these become exhausted is the metamorphism carried further and the less fusible rocks or rock-mixtures melted.

Many refractory rocks such as quartzites, clay slates, limestones, etc., obviously remain unfused, but it may often happen that fusion starts at the contact between two or more refractory rocks, and they may then continue to be melted down into a fusible mixture.

In a metamorphic region the most fusible mixtures afforded by the contacts are evidently formed first. After a while either the base or acid material becomes more or less used up and magma tending towards one of the extremes and at a higher fusion point and under greater stress has to be formed.

Finally the stress necessary for fusion at this center becomes so high that the adjacent regions of unmelted rock reach the fusion temperature and relieve the stress. The refractory residue of the old center of fusion then ceases to be fused any further.

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[12] Iddings, *Igneous Rocks*, p. 257 (1909).

[13] Barus, *American Journal of Science*, *4th Series*, vol. ix., No. 51, p. 173 (Mar., 1900).

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Evidence of Mountain Chains.—Mountain chains have a great degree of permanence. This can be explained on the assumption that underneath them are zones of weakness in the earth's crust which make it yield by plastic flow and fusion.

An obvious cause which can be assigned for such weakness is the heat generated by plastic flow. Such heat would render further plastic flow and fusion more facile, so that the mountain chain would support itself in one position in spite of rapid denudation or creep gradually in one or both of the directions at right angles to its axis.

Ocular Evidence.—In great bathyliths there is generally evidence that the rocks surrounding the igneous cores have been nearly in a state of fusion. These rocks are usually of gneissic or schistose character, showing much plastic flow and changing gradually into the igneous material.

It is rare on obviously intrusive igneous contacts not to find a very sharp line as between intrusive and wall-rock.

Evidence of Active Volcanoes.—As I have shown elsewhere,[14] the lava intruded in the regions where the horizontal stress exceeds the vertical, takes a horizontal tabular form and raises the superincumbent strata. These upper strata ultimately become convex enough[15] to overcome their excess of horizontal stress and the lava then breaks vertical rents upward for itself. As it gets nearer to the surface the aqueous vapors and other gases which it contains expand very greatly, and the column of lava is thereby so much lightened that the whole, or part, of the liquid contents of the horizontal cavity are rapidly extruded. The surface then subsides[16] again and the vertical openings are sealed. When a sufficient amount of lava has again accumulated in the laccolith form it is again extruded, and so on again and again.

In the absence of aqueous vapors and other gases, the pressure may be just enough to extrude the material slowly and regularly as "Fissure (Massive) Eruptions."[17]

The calculated position of the seat of a volcanic earthquake is not necessarily coincident with the place of formation of the

[14] *Trans.*, xli., 650 (1910).
[15] Geikie, *Text-Book of Geology*, 3d edition, p. 231 (1893).
[16] *Idem*, p. 244.
[17] *Idem*, p. 255.

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magma. The earthquake is more probably due to the above mentioned extrusive effects occurring at less depth.

Composition of Erupted Material.—The idea that igneous rocks are identical with ancient surface and intrusive formations is an old one, but geologists were not generally prompted thereby to conceive that bathyliths were not intrusions, but igneous sources.

The mean composition of the material issuing from a volcano is strikingly like that of the fusible sedimentary rocks. The lava corresponds to ordinary surface-rocks in fusible proportions. The steam corresponds to the water of their pores, the hydrocarbons to some of their coal and mineral oil; the chlorine closely suggests the sea-water which they often contain, and nitrogen suggests nitrates and dissolved nitrogen.

It therefore appears, according to this hypothesis, that gold and its siliceous rock associates may have been brought together at the surface as sand- and gravel-deposits, from which all the more easily decomposed minerals, which also happen to be the more base minerals, have been washed.

The same reasoning may apply to tin, though in a lesser degree, a more delicate adjustment of conditions being necessary for the deposition of this metal.

Silver, lead, and zinc, though in many ways associates of gold, also occur as constituents of base minerals which are easily slimed, and which, in the course of denudation, find their way to the sea, where they may settle with the fine sediments. These have usually a composition approaching that of the more base igneous rocks.

It is not as easy to explain the circulations of copper. Being an exceedingly soluble metal, however, it may find its way to the sea as a solution, being precipitated in very small quantities over vast areas.

Lime is precipitated in a similar manner, and its calcium is often partly replaced by magnesium.

This selective resemblance of copper and magnesium may explain the frequent association of the two elements in igneous rocks.

Sulphur, from the sulphides, having become a sulphate or sulphurous salt, is also very soluble, but is ultimately deposited,

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often as gypsum. From this state it is probably reduced by carbonaceous matter.

This carbon becomes buried in considerable quantities with the sediments, and probably induces the highly reducing nature of all eruptives and their emanations and the CO_2 which occurs in the latter.

It is well agreed that sediments have been brought down to depths where severe metamorphism goes on. It is only a minute step more to fusion.

The oldest rocks we know of as sediments do not go far beyond the history of life on the earth's surface. Before this time there must have been a vast period when water was too hot for life, atmosphere pressure was greater, and the physical commotion of land, water, and atmosphere enormous. It therefore follows that a vast amount of churning up of rocks, and incidental concentration of their minerals, must have occurred before the dawn of known stratigraphical history. These rocks are now doubtless fused or metamorphosed beyond recognition. Indeed, without life we know of no way by which they could be identified.

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Centrifugal Machines for Ore-Grading and Ore-Concentrating.

BY GODFREY T. VIVIAN, CAMBORNE, CORNWALL, ENGLAND.

(Cleveland Meeting, October, 1912).

VERY often important discoveries are made in one industry that may be used to advantage in another, but, owing to the rarity that men step out of one industry into another, these discoveries remain unknown outside of the trade or profession in which they are made. The present paper is written solely with the object of bringing to the notice of mining and metallurgical engineers a centrifugal ore-concentrator which eventually may have a great application.

Ore-Grading.

Fig. 1 represents the action which takes place when solids of different dimensions are stirred up in a box and allowed to settle as minerals of the same size but of different specific gravity. This principle is applied practically in grading china-clay and in enriching tin-slimes. In the latter case the slimes are stirred around in a tub with water and allowed to settle while at the same time the side of the tub is struck with a hammer. The colloidal slimes are then poured off and various slices taken off horizontally. The great drawback of this process is that the particles are deposited horizontally and all the colloids are lost.

Fig. 2 represents the action in which the particles are deposited side by side. Here the drawback is the great length of the box needed in order to prevent excessive loss. The alternative to lengthening the box is to increase the force of gravity. Centrifugal force may be regarded practically as the force of gravity augmented.

Fig. 3 illustrates the Gee centrifugal machine for grading ores. The drum, *A*, fitted with a box, *B*, is mounted on a shaft or spindle, *C*, the whole being suspended from a ball-bearing of special design at *D*, supported between girders, *E*. Rota-

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tion is imparted by the pulley, *F;* a band-brake is fitted at *G*. The upper end of the drum is closed by a cap, *H*, which makes a water-tight joint with the drum at *I*, when clamped by the lock-ring, *T*. (This forms a bayonet joint.) The cap, *H*, has a hole in the middle and is held central on the spindle by means of the casting, *K*, which is a sliding-fit on the spindle, and is connected with the cap by the upper ends of six rods, *L*. At the bottom of the drum, *U*, is fitted a weir-plate or diaphragm. Depending from the cap into the drum is a cage, seen best in the

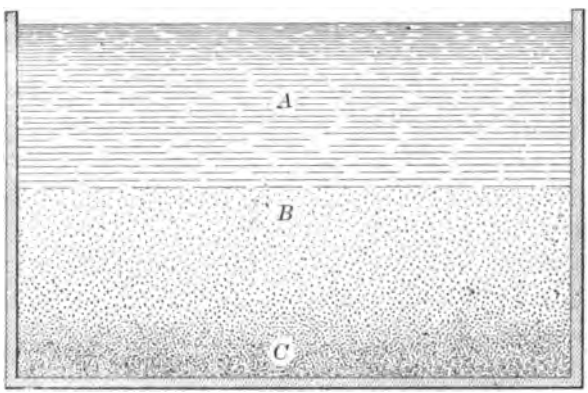

FIG. 1.—SETTLEMENT OF PARTICLES OF SAME SIZE BUT DIFFERENT SPECIFIC GRAVITY IN HORIZONTAL LAYERS.

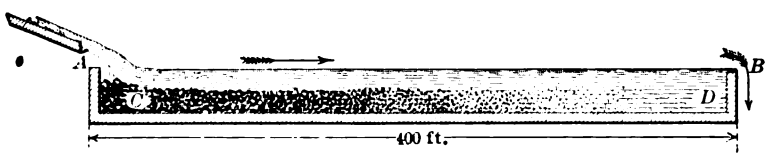

FIG. 2.—SETTLEMENT OF PARTICLES SIDE BY SIDE.

section, consisting of six vertical square rods, *LL*, to which are attached the radial blades or vanes, *N*. These blades extend the whole length of the drum, being connected to the cap at the top and to a circular plate, *O*, at their lower ends.

Fig. 4 shows the container clearly. This container slides easily on the drum, which it divides into six longitudinal compartments, each of which is provided with a curved plate. The container is in effect a movable lining to the drum, on which the recovered solids are received.

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Fig. 5 shows the manner in which the slabs are removed.

Fig. 6 shows the container in a raised position, either before or after removal of cake.

Fig. 7 shows the cake divided by imaginary lines for grading.

Apart from grading china-clays, pigment-material, graphite, etc., the question which every mining and metallurgical engineer will ask is, How will the machine behave with gold-slimes? The cake is evidently drier than the pulp falling from a vacuum-filter. So far as I am aware, the experiment has never been tried practically.

Ore-Concentrating.

The concentrating-machine is illustrated by Fig. 8.

A is mounted on the base, *B*, and closed at the top by the cap, *C*. Ball-bearings, *DD*, at the top and the bottom of the drum are placed so that the drum may rotate on, but independently of, the shaft or spindle, *E*. As this machine is run at very much lower speed than the grading-machine, the bearings are placed at both ends. The weir-plate at the bottom, inside the drum at *F*, serves the same purpose as in the grading-machine, namely, to provide an inner wall of water of the desired thickness within the drum. Attached to the shaft, *E*, are a number of vanes or blades, extending the whole length of the drum, and so arranged that the distance they dip into the wall of water may be varied, so that the grip of these blades on the water may be adjusted.

The principle of the operation is, that the drum is caused to rotate at a given speed and the wall of water is caused by the vanes to rotate within the drum at a greater speed, but in the same direction. These speeds and their ratio to one another may be adjusted within very wide limits.

The result is that the particles of greater specific gravity, for instance, tin oxide with a gravity of 6.7, are, by reason of their greater inertia, caused to deposit on the drum and remain there, while the particles of lesser gravity—let us say quartz, 2.68 (?)—are discharged. It will be seen from this that the force of gravity as generally operating in concentrating-machines is practically augmented 100 times. There are five different factors in adjusting the apparatus: (1) The speed of the drum. (2) The speed of the water. (3) The grip of the vanes (forced by moving them towards or away from the drum). (4) The thick-

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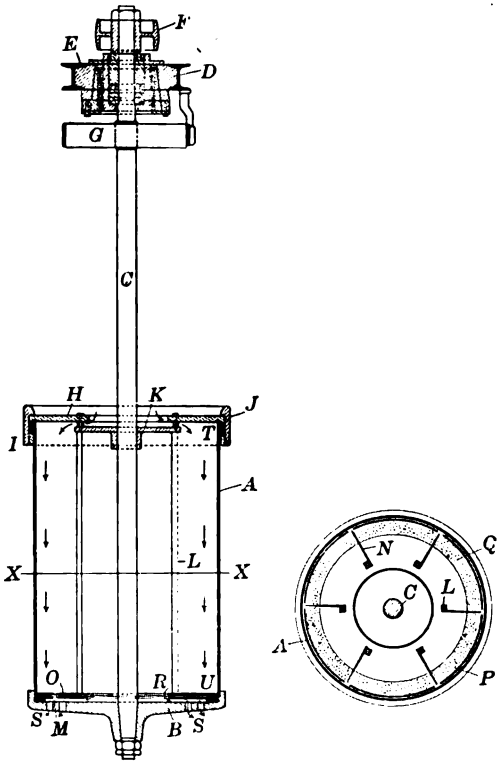

FIG. 3.—THE GEE CENTRIFUGAL ORE-GRADING MACHINE.

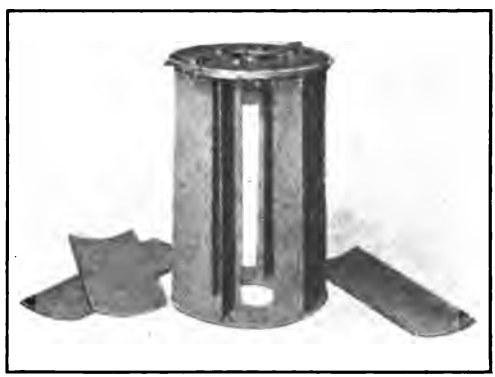

FIG. 4.—THE CONTAINER, OR DRUM-LINING.

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FIG. 5.—VIEW SHOWING REMOVAL OF SLABS FROM THE CONTAINER.

FIG. 6.—VIEW SHOWING CONTAINER IN RAISED POSITION.

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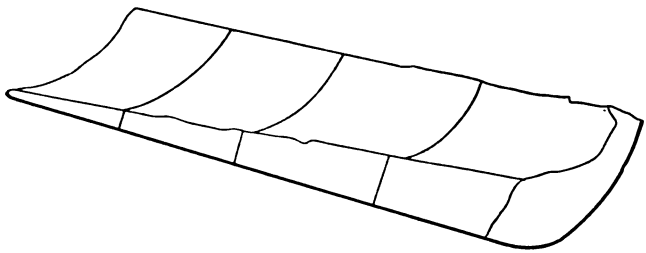

FIG. 7.—CAKE DIVIDED FOR GRADING.

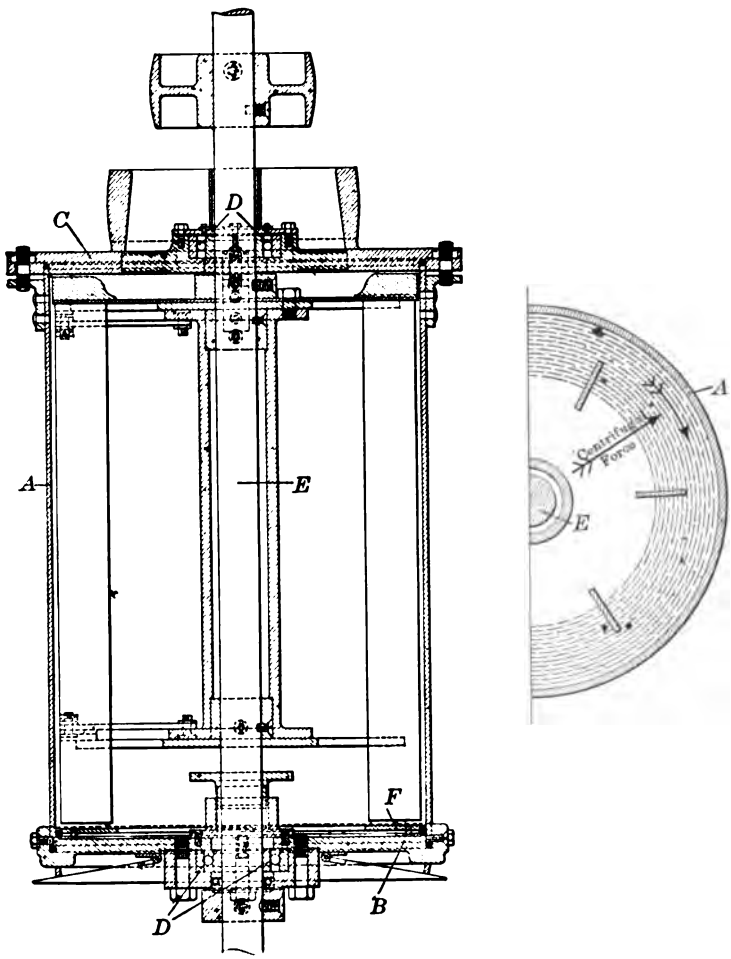

FIG. 8.—THE GEE CONCENTRATING-MACHINE.

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ness of the wall of water. (5) The rate of flow through the drum, which varies the time during which the material is being acted on.

It will be at once inferred that this is a machine to be used on very fine material. Laboratory-experiments have been made with tin-slimes, which in practice are absolute slimes—giving a total extraction of from 95 to 97 per cent. While the results of these experiments are better than those that can be obtained in practice, these centrifugal machines may offer a commercial solution to the present slimes-problems and are such as warrant a further investigation.

I beg to thank William Gee, Steinhoff Sons & Muir, Ltd., of London, for their courtesy in supplying information.

Discussion.

Courtenay DeKalb, Tucson, Ariz. (communication to the Secretary*):—Mr. Vivian's paper is interesting as an evidence of renewed attempts to solve the problem of adapting the centrifugal machine to the uses of the metallurgist. It would not seem that the Gee machine has solved the problem, because it is manifestly of use only with quite high-grade material, since low-grade pulp could not stand the cost of treatment by a discontinuous process. The mechanism, however, would seem to accomplish grading more perfectly than the ordinary centrifugal basket. This is an entirely different thing from concentration, which is intended to yield a finished marketable product. The real problem is to design a successful continuous centrifugal machine. Ten years or more ago a machine, which might be described as having an inverted step-basket, offered much promise, and was constructed and tested at considerable cost, but it presented insuperable difficulties.

The construction of a successful continuous centrifugal would be welcomed by the whole industrial world, since its uses would be most extensive. Mr. Vivian suggests the utilization of the centrifugal machine as a substitute for slime filters in the cyanide process, but he has apparently not tried the experiment. I tested the efficiency of centrifugal machines for this purpose in 1897–8, having to extract the cyanide solution from a highly

* Received Dec. 22, 1912.

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argillaceous slime of the sticky kind which the Mexican calls "lama." Our filter-presses yielded cakes containing usually 20 per cent. of moisture, and so dense as to preclude washing. The centrifugal machine readily brought the moisture content down to 9 per cent. and permitted fairly successful washing. Despite these advantages, the cost of power, labor, and repairs, and the loss of time in discharging the cake, exceeded the costs and demurrage when using the filter-press. The power required to operate a centrifugal machine is greater than those unfamiliar with its practical operation would imagine. It takes comparatively little power to maintain speed when running at normal velocity, but my recollection is, writing as I am without access to my notes, that a 54-in. machine requires 150 h-p. to start under load and accelerate up to normal velocity. Anything smaller than 54 in. in diameter would have too limited a capacity to be worth while, and it will readily be seen that a machine needing 150 h-p. on call must be made to run continuously to yield economic results.

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The Wood Flotation Process.

BY HENRY E. WOOD, DENVER, COLO.

(Cleveland Meeting, October, 1912.)

In my opinion, the concentration of minerals by flotation is the most interesting problem in ore-dressing, and will command eventually far more consideration than it has at present. For many ores it furnishes a complete method of concentration absolutely independent of specific gravity, and thus simplifies the milling of many difficult combinations. Not having to recognize the influence of specific gravity, we are also able to dispense with some of the complications resulting from classifying-conditions.

Where flotation cannot be used for a complete separation of the minerals from the gangue, it will be found, in nearly all cases, to save those portions which are generally lost by reason of their fine state of division. For this reason, it should be looked upon mainly as an aid to general concentration, although there are many ores so well adapted to flotation that more than 90 per cent. of the valuable metals can be recovered from them in one operation by this process.

In presenting the results of my investigations in flotation, I have endeavored to theorize as little as possible. That side of the subject has been extensively discussed by others.

My earliest work with flotation was done in 1895, when I used various oils, grease, soap, etc., with results that were remarkable. I did not realize then that with water alone equally good separations could be made. Up to 1906, my endeavors were confined to attempts to keep the concentrates on the tables well wetted and away from exposure to the air. In common with others, I observed that when there was any obstruction in the wash-water on the table of sufficient size to divert the water to either side, the concentrates collected, and, on exposure to the air, would promptly float away. This convinced me that the sulphides were all good "swimmers," and that if,

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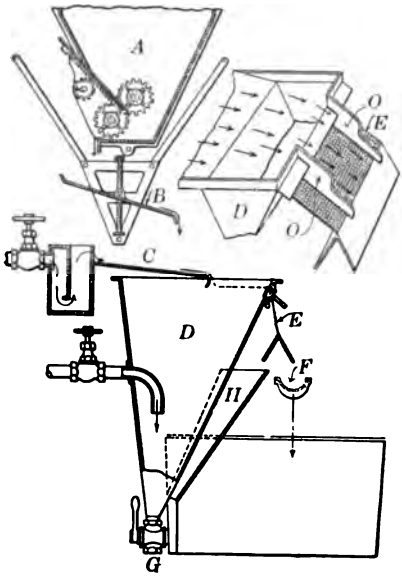

Fig. 1.—Type *A*.

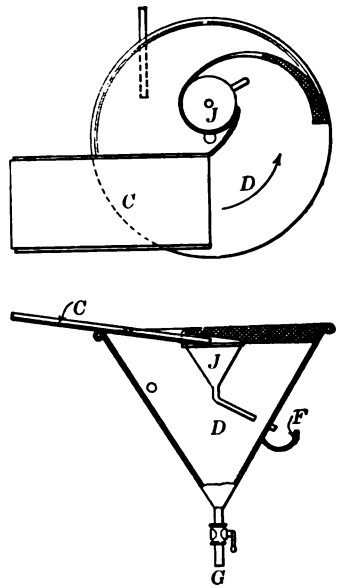

Fig. 2.—Type *B*.

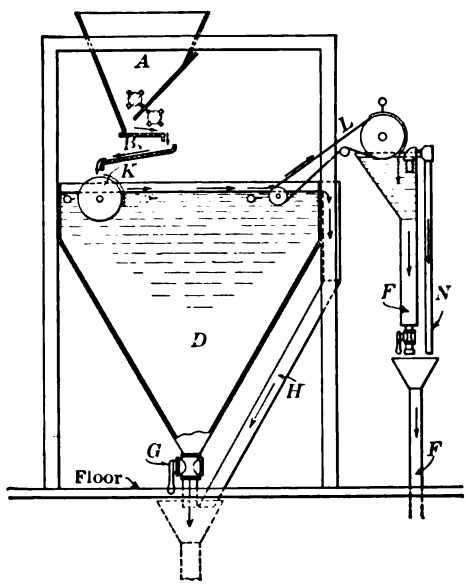

Fig. 3.—Type *C*.

FIGS. 1, 2, AND 3.—THREE TYPES OF THE WOOD ORE-CONCENTRATING MACHINE.

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instead of endeavoring to drown them, they were given a chance, they would all float.

Having successfully demonstrated this fact, as well as another equally important, namely, that practically all the oxides are quickly wetted, become saturated like a sponge, and, sinking under the surface-film, become subject to gravity-conditions, I commenced my experiments.

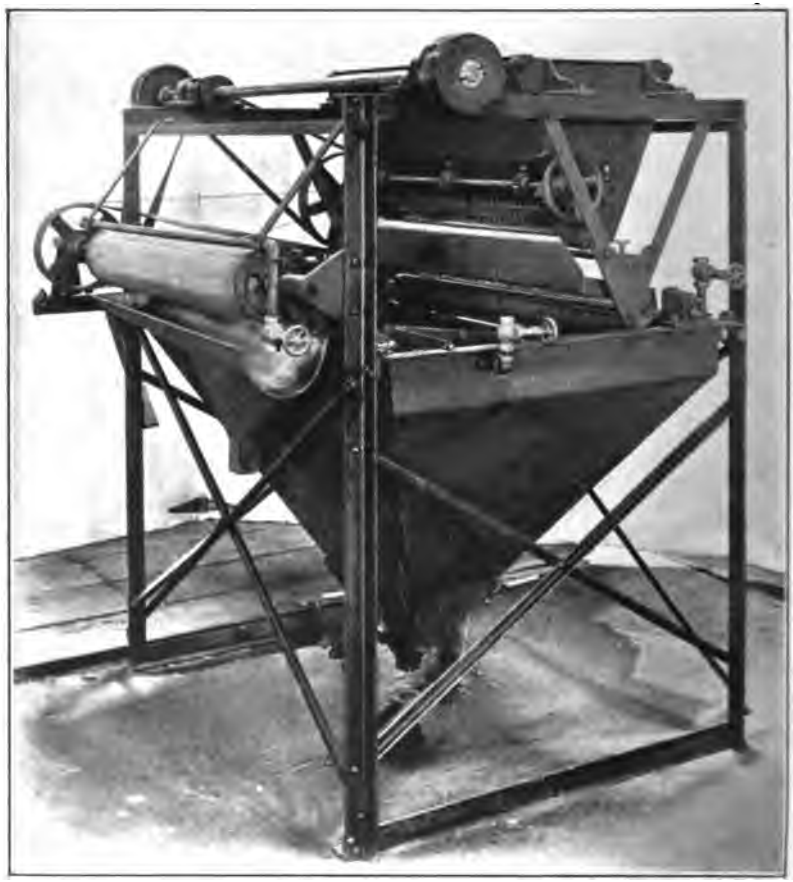

FIG. 4.—PHOTOGRAPHIC VIEW OF THE WOOD ORE-CONCENTRATING MACHINE, TYPE *C*.

I found that if dry-crushed ore, not necessarily very fine, was gently deposited upon a swiftly-moving sheet of water, separation of the sulphides from the gangue took place. Retarding the current permitted the gangue to sink, while a film of sulphides remained on the surface. To collect this film I

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have devised three types of machine, shown in Figs. 1, 2, and 3. Fig. 4 is a photographic view of the type *C* machine.

All three types are similar so far as the system of feed is concerned. In type *A*, a sectional view of which is presented in Fig. 1, the unsized ore, at whatever mesh has been decided upon, and ranging from 10- to 40-mesh or finer, is delivered to a feed-hopper, *A*, the outlet from which is under close control, so that a uniform amount of ore can be steadily dropped upon a plate, *B*, vibrating 500 times per minute. From this plate the ore falls direct to the sheet of water, which flows over a plane, *C*, inclined at 7°. The film of sulphides crosses the tank, *D*, and in falling over the outlet is dewatered and split off by means of a nearly vertical screen, *E*. *O*, *O*, are guides or stretchers, and perform quite an important function when this type of machine is used. The elastic properties of the film are such that if these guides were not used, the sheet of water emerging from the tank would quickly assume a spiral shape, which would result in a wetting of the sulphides. These would then pass through the screen instead of over it.

The guides are so adjusted that the water intersects the screen in a line parallel to the surface of the tank. By slight adjustments one is then able to split off cleanly the film of concentrates from the water and the gangue in suspension. A separate patent was allowed for this feature. The water passing through the screen, containing more or less fine gangue in suspension, is caught at *H* and conducted to a launder, *F*, which also collects such coarse mineral and gangue as steadily issues from the bottom outlet of the tank, *G*, from which it passes to tables. I sometimes find it advantageous either to classify hydraulically or screen-size the tank contents *en route* to the table; but it is not essential in all cases, because, having removed the fine sulphides, as well as a large proportion of the larger sizes, by flotation, only the coarser mineral and the coarser gangue have to be considered, as the fine gangue in suspension passes off with the wash-water.

The feed of the type *B* machine, Fig. 2, is delivered over the inclined plane *C* to a conical tank, *D*, in which a rapidly-revolving motion with vortical effect is given to the surface of the water by means of a water-jet against the side of the tank. The feed falls upon an inclined sheet of water. The film of sulphides

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is guided by means of partly-submerged vertical screens, *E*, to a funnel, *J*, the outlet of which penetrates the side of the tank, where the concentrates are delivered to a launder, *F*.

The output of a tank 5 ft. in diameter will exceed 0.5 ton per hour on a 20-mesh ore that is adapted to flotation. Necessarily, a small amount of water is required to carry the film into the funnel. The film, having a definite thickness, is probably carried on an amount of water of two or more times the volume of the concentrate. As this water carries more or less gangue in suspension, it makes the concentrates siliceous. In certain sections of Mexico, where copper sulphides are in demand and silica is not objectionable, the conical-tank plan meets the requirements of the situation; but this form would not be satisfactory, for instance, in Utah, where a high-grade copper-concentrate is of more importance than the highest possible recovery. In Utah there is a penalty imposed by the smelters for silica above a certain amount, a condition which does not exist in Mexico.

In the type *C* machine, Fig. 3, the 20- to 40-mesh ore is dropped from the vibrating plate, *B*, to a corrugated rubber- or canvas-covered cylinder, *K*, which is revolving in water. Enough water is elevated above the water-level of the tank to wet the gangue and float the sulphides. This sulphide film passes from the cylinder over the tank-surface to an endless canvas belt, *L*, which emerges from the tank and is controlled by sprockets. Water-jets gently force the concentrates upon the canvas, but, as the canvas is elevated, the water drains back to the tank, carrying with it any gangue in suspension. The concentrates are deposited in another tank and collected in a launder, *F*. An automatic siphon, *N*, prevents an overflow of concentrates from the tank after they are discharged from the belt. *G* is the outlet-valve controlling the discharge of tailings after flotation. In most cases the tailings pass to a Wilfley table. Quite frequently, however, the recovery made by flotation makes any additional treatment unnecessary.

As clean wash-water is used at the delivery end of the take-off belt to remove the concentrates, they have a minimum silica-content. The difference between type *B* and type *C* in this respect was shown by comparative tests. The crude ore used in each test assayed 2.63 per cent. of copper. The *B* machine

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saved 88 per cent. and gave concentrates assaying 18 per cent. of copper, with 23 per cent. of SiO_2, while the *C* machine saved 93 per cent., giving concentrates assaying 27 per cent. of copper, with 3.6 per cent. of SiO_2, both tests being made by flotation alone.

As the corrugations of the rubber-covered cylinder are only adapted to comparatively fine ore, say 20-mesh and finer, they would not bring up enough water to saturate particles that were larger than the corrugations; hence, in treating 8- or 10-mesh ore, all of such particles would not reach the water, and other dry ore would fall on top of them, causing an unclean concentrate. To prevent such conditions in treating coarse ores, the cylinder is removed, and a rapid impetus is given to the surface of the water by means of water-jets from a pipe just in advance of the position formerly occupied by the cylinder.

An ore suitable for jigging, in which the sulphides are liberated at 4-mesh, can be fed, either sized or unsized, direct to the tank. The treatment of coarse ore, say from 4- to 12-mesh, is performed for the purpose of floating off fine particles of sulphides which mechanically adhere to the coarser sizes. I have carefully dry-screened ore from 4- to 20-mesh and then subjected these coarse sizes to flotation, the result being a high-grade product that would add 5 per cent. to the general recovery.

During my experimental work on more than 800 separate samples, many interesting incidents were observed, among which I note the following:

If, after grinding a given ore to the right size, it is at once flotated, from 30 to 85 per cent. by weight of its sulphide particles are likely to float. If another portion, prepared at the same time and under the same conditions, is set aside for several days and then treated, less than one-third of what floated before will remain on the surface. This applies particularly to the copper sulphides, and must be due to partial oxidation, although such a change is not visible to the eye. This behavior of copper-ores is more pronounced if steam-heat be applied in drying a previously-made concentrate for flotation. It is therefore not advisable to regrind wet jig-tailings and dry them for flotation. To secure the best possible flotation from copper-ores,

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they must be ground directly to the required mesh for separation of the mineral from the gangue, and promptly milled.

Curiously enough, this state of affairs does not exist in the treatment of some other complex sulphides, for instance, those containing galena and the non-magnetic resin blende. At 40-mesh a 10-per cent. lead-ore, with 20 per cent. of zinc, will produce a flotation-concentrate accounting for 80 per cent. of the lead, assaying from 56 to 60 per cent. of Pb and 7 per cent. of Zn. If the tailings, after flotation, are passed over a Wilfley table, a high-grade coarse lead-concentrate is easily cut out from the zinc and silica. The removal of the silica from the zinc-middlings enables us to obtain a high-grade zinc-concentrate. Now, if this blende, which sank in the tank during flotation, be dried and re-treated by flotation, nearly all of it will float, while the accompanying silica will sink, leaving a high-grade product. This particular effect is not so pronounced when the blende is of the magnetic or so called black-jack variety.

I have noted in the treatment of copper-ores that the tendency to float decreases as the sulphur-content diminishes. More than 90 per cent. of the copper of chalcopyrite ore with a hard white quartz gangue can be saved with less than 5 per cent. of silica in the concentrate by flotation alone; but if bornite or chalcocite is present, the proportions of these minerals saved are much smaller than of chalcopyrite. If chalcocite contains a small percentage of chalcopyrite, practically all of the latter will float and some of the fine chalcocite with it. This leaves the coarse chalcocite in good shape for final table-separation.

Most of the magnetic minerals will sink, so that a good flotation-concentrate can sometimes be made from a mixture of chalcopyrite associated with a magnetic blende. Magnetite will sink and leave suspended on the surface any fine sulphides that may be with it, as well as the metals, such as gold, silver, platinum, etc. To obtain the best effects for the fine metals, we sometimes modify the film by using a minute amount of oil. The oil being used purely as a conveyor, and not as an enveloping element, my patents do not conflict with the oil-systems. My main idea is the use of water alone. I rarely use oil and never acid or gas.

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Judging from the appearance of a 20-mesh flotation-concentrate, the inexperienced observer assumes, from the smooth surface of the film, that only the extremely fine sulphides are being recovered. A screen-analysis of the concentrate, however, shows that more than 10 per cent. of the concentrate is between 20- and 40-mesh. Apparently the elasticity of the film is such that it supports the coarser particles, but not rigidly, so that only the apex of each piece of sulphide is shown, and as it lies along with the fines, a uniform, smooth surface is presented.

The strength of the film is illustrated in another way. If particles of dry ore are placed upon the film of sulphides, they will be thrown off by the rebound of the film as it passes over the curve of the dam, the effect being practically the same as the repulsion of highly-conductive minerals in passing over an electrostatically-charged roller.

The elastic properties of the film are plainly shown during the operation. The sulphide film spreads out rapidly over the tank-surface. If its passage is restricted, it will immediately be compressed into ridges, which in time become sufficiently heavy to sink of their own weight. Also, before sinking, the film will creep up vertically on the corrugated roller.

Violent agitation of the film must be avoided, or the particles will become wet and sink. Nevertheless, the tenacity of the film is remarkable.

As long as the feed is continuous, the sulphides on the film easily slip over to the take-off belt. This action is assisted by a gentle water-pressure directly in front of the belt. A gentle current of air also produces the same result.

I have alluded to oxidizing influences preventing or interfering with flotation-results in the treatment of copper sulphides. With several other sulphides this is not the case, so that flotation can frequently be applied, after amalgamation and concentration, as a more economical method of recovery than cyanidation of the tailings.

For instance, I had a $20 gold-ore, of which $15, or 75 per cent., was removed by amalgamation and concentration. Of the $5 left in the tailings, 93 per cent. was extracted by cyanide, but at a cost of $2.75, by reason of the high cyanide-consumption, caused by values still locked in the fine sulphides. The

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same tailings, assaying $5, were dried and flotated, producing a 10-oz. gold-concentrate and leaving 30 cents per ton in the final tailings. The cost was far less than that of cyanide treatment.

Nearly all the average free-gold ores of California contain arsenical pyrites, and this, in turn, carries gold that is not free. Such ores, if crushed dry and treated by flotation, yield high-grade concentrates, accounting for from 10 to 20 per cent. of the total valuable metals in the ore. After flotation, the free gold can be amalgamated as usual. I have found that all that is necessary is to feed over an amalgamated plate, which has an inclination of 7° and is covered by water.

Such flotation-concentrates of arsenical pyrites frequently assay many thousands of dollars per ton, and constitute an important item in the total recovery. A considerable percentage of the values in some varieties of wet-crushed ores, particularly of copper and arsenical sulphides in a hard gangue, will float. While we cannot do anything with sulphides in suspension, we can readily save all floating slimes from wet-crushing by this same general system, and with a simple device placed in any launder below the mill, provided the surface-conditions are not rough.

All over the world, valuable ores, such as sulphides of silver, gray copper, galena, etc., are frequently found in a matrix of baryta, or associated with heavy garnet. The specific gravity of such gangues practically prohibits either jigging or table-separation; but when this method of ore-dressing is applied to them, from 80 to 90 per cent. of their values can be recovered at one simple operation. A recent example of this was a 1,200-oz. silver-concentrate made from a Mexican ore containing 78 per cent. of $BaSO_4$. Another, from Canada—a crystallized barite containing 10 per cent. of Pb—gave a 90-per cent. recovery in a 70-per cent. lead-product. Many mines containing ores of such composition have been abandoned because of the closeness of the specific gravities.

We have had several ores of nearly clean pyrrhotite containing a small percentage of chalcopyrite. Most of the latter will float by itself. If the remainder is concentrated, the ratio of concentration is too low to make a valuable copper-product; but if this concentrate is dried and passed through a magnetic separator, such as the Wetherill or Dings, at a low amperage, the pyrrhotite is lifted off and the non-magnetic tailings will be

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a clean chalcopyrite. This, combined with the flotation-concentrate, gives a total high extraction.

The capacity of the series of flotation-machines here shown is variable, depending upon several conditions, among which are the nature of the gangue, the size of the ore, and the concentration-ratio. A standard machine treating a 20-mesh quartz ore, using a 3-ft. width of feed and having a 4-ft. take-off belt, will vary in capacity from 1,000 to 2,000 lb. per hour, unless the ratio of concentration is low, in which case the capacity will be smaller. Some ores that possess an easily wetted gangue and call for a high concentration-ratio can be fed rapidly at 20-, 30-, or 40-mesh.

For instance, a 1 or 2 per cent. molybdenite ore in a quartz gangue will give a clean concentrate, even if the ore is fed several times faster than an ordinary sulphide ore; and the same is true of graphite. In a special form of the machine, designed for the concentration of molybdenite and graphite, one or two partly-submerged cylinders, covered with either canvas or fine wire screen, are inserted in the tank. These cylinders revolve in the same direction and at the same rate of speed as the main take-off belt or belts. The film-concentrate is lifted from the tank, and passes over the rollers down to the tank-level again. With each elevation partly-saturated gangue becomes thoroughly wetted, and drops off into the tank without any apparent disturbance or wetting of the concentrate. Such action naturally raises the grade of the concentrate. In leaving the take-off belt, it is not washed off by direct water-jets, as is done with other ores, but the take-off belt is allowed to discharge its load at water-level in a shallow compartment, and the concentrate gently floats off to a collecting-launder.

In the flotation-treatment of molybdenite it frequently happens that copper sulphides, iron pyrite, and other impurities, are present, and must be removed to make a salable product. If copper and iron are present, they are re-concentrated on a Wilfley table, so handled that the tailings will contain the molybdenite. Another way is to dry the flotation-concentrate, give it a gentle roast, and pass it through a magnetic separator, to remove the copper and iron. Molybdenite- and graphite-concentrates are frequently contaminated by mica. This in some cases can also be taken out by the magnetic separator,

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or by re-flotation with the water at 110° F. It can also be removed electrostatically.

Objections are made to the drying of flotation-concentrates on account of their supposed fineness, but this is a mistake, as mentioned previously. There should be no more difficulty in drying concentrates of this nature than those produced by vanners or ordinary slime-tables. By alternating settling-tanks, which can be siphoned, or by using filter-presses, they can be constantly dewatered. They can be briquetted if necessary.

There are, in Mexico and elsewhere, dry-crushing mills that use magnetic separators to remove large amounts of gangue in the shape of garnets, siderite, etc., in order to get more favorable conditions for gravity-separations on the tables for the lead and zinc. If a flotation-machine be placed so that it will take the non-magnetic tailings before they reach the table, a high-grade lead-product results, and aids in the general separation of the coarse lead and zinc, improving each from 15 to 20 per cent.

Flotation will be found to be of the greatest benefit when applied to ores that have the largest losses in milling, as well as to certain ores that are virtually not subject to wet concentration on account of small differences in specific gravity.

The proper preparation of ores for flotation is essential. For instance, if a small sample is pulverized unnecessarily fine, in such a state as would result from the use of an ordinary muller, poor returns may be expected, but if the same ore is shattered by pounding on an iron plate, most favorable products will ensue, as the crystals retain more of their original shape.

For this reason, I have found that ores should be prepared in ball-mills, or by rolls, rather than in any machine that grinds much of the gangue to an impalpable dust, in which condition both the gangue and the mineral of many ores will float. Flotation-treatment of definitely-sized ore will materially increase the capacity of the large sizes by themselves, but mechanical difficulties are encountered in feeding the smaller sizes by themselves, and the capacity is also much reduced; hence it is advantageous, after having decided upon the proper mesh to liberate the sulphides from the gangue, to feed them all at once without previous sizing.

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I wish, also, to place special emphasis upon the property of easy saturation or wetting of nearly all of the oxides. The only one that has surprised us and floated in appreciable proportions is the specular hematite in certain New Mexico zinc-ores, which floats up to 10 or 20 per cent. by weight. This is largely due to the tabular shape of its crystals. After magnetic removal of the iron, the non-magnetic tailings assay from 45 to 50 per cent. of zinc.

Simple as these flotation-devices are, it has taken several years to secure patents, both domestic and foreign, as well as to develop the general standard type of machine now on the market. It weighs 1,000 lb. net; measures 4 by 5 by 6 ft. Ready for mule-back transportation, its weight is 1,100 lb. It requires 0.25 h-p. and less water than is used in wet concentration.

Sulphides of the following metals are particularly favorable for flotation methods: Copper, lead, silver, antimony, bismuth, mercury, arsenic, molybdenum, zinc, iron, nickel, etc., as well as tellurides, graphite, gold, and platinum.

When tellurides are present, a high-grade flotation-concentrate is obtained and the tailings are left in excellent shape for cyanidation. I have proved this commercially on ores from Boulder county, Colo.

Results on Molybdenite.

The concentration of molybdenite by flotation is decidedly interesting, in that it furnishes illustrations of several of the peculiarities of surface-tension. For instance, in the recent flotation of a Canadian ore of this nature, with a quartz-schist gangue, which assayed 1.87 per cent. of MoS_2, I obtained an apparently pure film of MoS_2, which, when assayed, was found to contain only 45.31 per cent. of MoS_2, and 41.97 per cent. of insoluble residue. On examination, the under side of the film was seen to be composed of fine mica scales, which must have adhered to the flakes of MoS_2. My first conclusion, that the mica was simply enveloped in a coating of MoS_2, was not correct. The recovery in this particular case was 79.08 per cent. The test was made with water having a temperature of 56° F.

The above examination was followed by that of an 8.65-per cent. molybdenite ore from Alaska. This was a complex ore,

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containing pyrrhotite and magnetite, and low in silica. The flotation-concentrate, at 40-mesh, amounted to 23.4 per cent. by weight. As it assayed only 30 per cent. of MoS_2, a recovery of 79 per cent., it was evidently contaminated by other minerals that had floated.

Another sample of the same ore, lightly roasted, gave a flotation-concentrate assaying 81.45 per cent. of MoS_2, the ratio of concentration being 35.71 : 1, but the saving was only 26.3 per cent. This indicates that either some of the molybdenum is present in the original ore as an oxide, or that the roasting was carried too far and oxidized it. In the first test the saving was 79 per cent., assaying 30 per cent. of MoS_2, while the second trial recovered but 26 per cent., assaying 81.45 per cent. It is possible, therefore, that by obtaining the right roasting-conditions, such an ore could be profitably treated.

The concentrates from one test of this ore assayed 1.86 per cent. of copper. After a gentle roast, this was removed by the Wetherill separator, leaving only a trace of copper, and with no material loss of MoS_2.

A certain Alaskan ore recently treated by me in commercial quantities furnishes an illustration of the extreme sensitiveness of chalcopyrite to oxidizing influences. It contained 4 per cent. of copper, closely associated with a heavy pyrrhotite in a siliceous gangue. Careful work on this ore by jigging, tabling, regrinding of jig-tailings, all followed by canvas slime-table treatment, gave a 15 per cent. copper-concentrate, with a recovery of 62 per cent. Direct table-concentration made a 16 per cent. copper-concentrate, saving 47 per cent. The same ore, by flotation alone, made a 25-per cent. copper-concentrate, accounting for 69 per cent. of the copper. To this was added 11 per cent. from the table-work, a total of 80 per cent. In making this examination the ore was dry-crushed at 8-mesh, and the from 8- to 20-mesh product (dry-screened) was passed through the flotation-machine, resulting in a high-grade concentrate, which added 4.25 per cent. to the total recovery. This proportion came from the mechanically adhering fine sulphides, and would be lost in ordinary jig-work. The from 8- to 20-mesh jig-tailings of this test were dried and reground for flotation; but on account of the oxidation not a particle floated.

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Screen-Analysis of 20-Mesh Flotation-Concentrates.

	Mesh.	Percentage by Weight.
No. 1	20 to 40	14.00
No. 2	20 to 40	10.00
No. 3	20 to 40	41.75

Many similar examples could be cited showing that flotation largely reduces the proportions going to the tables, thereby increasing the efficiency of the latter.

Table I. gives the results obtained by flotation alone. All of the ores tried were difficult ones to treat, and several of them were not susceptible to gravity-separation. The recovery by flotation alone, in many instances, exceeded the returns from the best possible treatment in the wet way.

TABLE I.—*Results Obtained by Flotation.*

Locality.	Mesh Size.	Assay of Ore.	Concentrations.	Assay of Flotation-Concentrates.	Recovery by Flotation (Concentrates).	Total Recovery.
Copper-Ores.		Per Cent.	Tons Into 1.	Per Cent.	Per Cent.	Per Cent.
Canada. Chalcopyrite	40	Cu. 2.60	8.25	17.10	79.72	94.48
Mexico. Chalcopyrite	40	Cu. 5.80	3.28	13.20	69.40	88.45
Colorado. Chalcopyrite	40	Cu. 1.42	5.97	5.40	63.75	83.29
Colorado. Chalcopyrite	40	Cu. 2.10	4.75	12.25	96.28	98.45
Canada. Chalcopyrite	40	Cu. 2.75	9.38	13.45	52.15	86.05
Colorado. Chalcopyrite	40	Cu. 2.50	7.04	15.00	71.59	88.35
Nevada. Chalcopyrite and garnet	40	Cu. 3.85	9.80	15.45	40.94	91.21
Arizona tailings	40	Cu. 0.47	28.25	4.80	43.95	74.16
Mexico	40	Cu. 15.20	2.14	26.50	81.48	89.01
Montana	40	Cu. 1.90	.....	11.40	42.92	90.45
Colorado	40	Cu. 8.28	.....	21.60	52.71	91.57
Gold-Ores.		Oz. Per Ton.		Oz. Per Ton.		
Colorado	40	Au 0.26	26.66	6.32	91.19	91.19
Mexico	40	Au. 0.42	5.83	1.00	40.98	94.41
Colorado	40	Au. 0.66	5.97	1.42	36.04	86.48
Colorado	40	Au. 3.50	4.75	4.86	29.24	89.62
California. Arsenical pyrite	40	Au. 90.96	14.59	311.00	28.43	94.52
Arizona	40	Au. 0.12	28.25	0.98	35.13	80.12
Silver-Ores.						
New Mexico	40	Ag. 15.90	23.25	184.90	50.01	90.01
Mexico	40	Ag. 95.60	5.83	340.90	61.11	86.26
Colorado	40	Ag. 3.00	5.97	10.60	59.12	88.61
Colorado	40	Ag. 12.96	4.75	43.30	70.34	84.65
Lead-Ores.		Per Cent.		Per Cent.		
Colorado	40	Pb. 12.20	11.11	54.00	89.85	90.86
Colorado. Baryta	40	Pb. 16.50	6.06	41.60	41.60	95.82
Canada	40	Pb. 19.70	5.35	59.60	56.55	79.44
Mexico	40	Pb. 5.60	5.83	16.90	51.77	89.65

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Copper-Ores.

Alaska.—30 mesh, 4.08 per cent. of copper. Close classification, jigging, table-concentration, with regrinding of jig-tailings, gave 65 per cent. of the copper, assaying 15.66 per cent. of Cu. Ratio, 5.9 tons to 1.

By flotation at 30-mesh—the same ore—the concentrate assayed 25.03 per cent. of Cu, recovering 73 per cent. at a ratio of 8.4 into 1. The Wilfley concentrate added to this 7.31 per cent. more, assaying 6.32 per cent. of Cu, ratio 21.19 to 1, a total recovery of 80.31 per cent. When treated at 40-mesh by flotation alone, the recovery exceeds the above total.

Colorado.—40-mesh.

	Copper-Assay. Per Cent.	Recovery. Per Cent.
Crude, 100 per cent. by weight	2.68	
Concentrates, 4.75 tons to 1	12.25	96.23

Silver-Ore.

South America.—30-mesh.

	Assay. Oz. Per Ton.	Recovery. Per Cent.
Contents	36.20	
Flotation-concentrates, ratio 3.64 to 1	106.90	80.90
Wilfley concentrates, ratio 16.01 to 1	36.00	6.19
Total at 2.97 to 1, assaying (average)	93.70	87.09

A second test of this ore gave 91 per cent. of its gold-content, 96.3 per cent. of the silver, 93.9 per cent. of the lead, and 84.41 per cent. of the copper. Several other combination methods applied to this ore failed to produce satisfactory commercial products.

Mexico.—20-mesh.

	Assays.		Recoveries.	
	Silver. Oz. Per Ton.	Lead. Per Cent.	Silver. Per Cent.	Lead. Per Cent.
Crude	36.5	9.00		
Flotation-concentrates, 17.4 per cent.	174.70	40.75	83.40	79.00
Wilfley concentrates, 8.34 per cent.	27.90	7.80	6.36	7.25
Total			89.76	86.25

The ore from Mexico when treated most carefully by wet-concentration gave 75.22 per cent. of the silver and 74.38 per cent. of the lead, both silver and lead being of much lower grade.

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Mexico.

	Silver-Assay. Oz. Per Ton.	Recovery. Per Cent.
Silver sulphide in quartz	38.2	
Direct table-concentration at 20-mesh, ratio 20.97 tons to 1	395.6	49.38
The same ore treated by flotation alone—9.09 tons into 1	323.0	93.02

Bismuth-Ore.

Colorado.

	Bi. Per Cent.	Recovery. Per Cent.
Crude, 100 per cent. by weight at 40-mesh	1.00	
Flotation-concentrates, 3.84 per cent. by weight	21.45	55.52
Wilfley concentrates, 9.34 per cent. by weight	3.25	34.77
Total concentrates, 7.52 tons into 1	6.79	90.29

Gold-Ore.

Nevada.

	Assays. Au. Oz. Per Ton.	Assays. Ag. Oz. Per Ton.	Recovery. Au. Per Cent.	Recovery. Ag. Per Cent.
At 20-mesh, 100 per cent. by weight	5.52	6.50		
Flotation-concentrates, 1.15 per cent. by weight	73.76	122.80	15.36	21.72
Wilfley concentrates, 7.90 per cent. by weight	43.26	48.20	61.96	58.62
Total concentrates at 11.04 to 1	48.87	57.58	77.31	80.34

Left in the tailings—$25 in gold and 69 cents in silver, in excellent condition for cyanide treatment, the sulphides having been removed.

Telluride Ore.

Colorado.

	Assays. Au. Oz. Per Ton.	Assays. Ag. Oz. Per Ton.	Recovery. Au. Per Cent.	Recovery. Ag. Per Cent.
Crude ore	2.10	3.60		
Flotation-concentrates, 2.2 per cent. by weight	36.50	52.40	37.24	32.14
Wilfley concentrates, 6.84 per cent. by weight	11.14	12.70	36.33	24.63
Total concentrates, 11.06 tons into 1	17.31	22.36	73.57	56.77

With extraction of the tellurides the values remaining in the tailings were successfully cyanided.

Zinc-Ore.

Utah.

	Zn. Per Cent.	Recovery. Per Cent.
Slimes	24.50	
Flotation-concentrates, 3.59 tons into 1	48.20	59.67

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Antimony-Ore (Stibnite).

	Sb. Per Cent.	Recovery. Per Cent.
Crude, 100 per cent. by weight at 40-mesh	3.57	
Flotation-concentrates, 6.50 per cent. by weight	47.40	86.30

Mercury-Ore (Cinnabar).

	By Weight. Per Cent.	Hg. Per Cent.	Per Cent.
Crude	100.00	6.30	
Flotation-concentrate	9.33	28.10	Recovery 41.58
Wilfley concentrates	4.33	76.90	Recovery 52.75
Tailings	86.33	0.40	Contain 5.48
Assay of combined concentrates, 7.32 tons into 1		43.56	Recovery 94.33

Molybdenite Ore.

Colorado.

	MoS_2. Per Cent.	Recovery. Per Cent.
Crude, 40-mesh	2.08	
Flotation-concentrates, 5 per cent. by weight	41.38	99

Re-treatment would materially increase the grade with little loss of MoS_2.

Alaska.

	MoS_2 Per Cent.	Recovery. Per Cent.
Crude, 20-mesh	2.00	
Flotation-concentrates, 3.33 per cent. by weight	61.55	92.27

Canada.

	MoS_2 Per Cent.	Recovery. Per Cent.
Quartz and mica gangue	1.87	
1*st Test*, H_2O at 56° F. Flotation-concentrates, 30.64 tons to 1	45.31	79.08
2*d Test*, H_2O at 110° F. Flotation-concentrates, 29.41 tons to 1	56.70	89.17

Foreign.

	Mesh.	MoS_2 Per Cent.		By Weight. Per Cent.	MoS_2 Per Cent.	Recovery. Per Cent.
(1)	20	6.66	Flotation-concentrates	10.5	63.50	96.34
(2)	30	6.73	Flotation-concentrates	8.6	67.42	86.26
(3)	40	6.66	Flotation-concentrates	8.8	65.50	86.48
(4)	40	8.95	Flotation-concentrates	11.29	74.60	94.19

Graphite.

Canada.

	Mesh.	Carbon. Per Cent.		By Weight. Per Cent.	Carbon. Per Cent.	Recovery. Per Cent.
(1)	40	19.46	Flotation-concentrates	25.55	69.38	91.27
(2)	40	65.34	Flotation-concentrates	72.00	79.92	88.12
(3)	40	6.30	Flotation-concentrates	6.80	71.14	76.79

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Silver-Ore—Gray Copper—Galena—Baryta.

Colorado.

	By Weight. Per Cent.	Assays. Silver. Ounces.	Assays. Lead. Per Cent.	Recovery. Silver. Per Cent.	Recovery. Lead. Per Cent.
Crude, 20-mesh	.....	5.2	13.2	.....	.....
Flotation-concentrates	18.4	21.5	58.3	76.2	81.3
Wilfley-concentrates	6.9	10.0	20.7	13.3	10.8
	25.3	18.4	48.0	89.5	92.1

Blast-Furnace Flue-Dust.

Pennsylvania.

	By Weight. Per Cent.	Iron. Per Cent.
Flotation-concentrates (coke)	11.8	.....
Tailings (iron)	85.0	64.60
Loss	3.2	.....
	100.00	

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An Experience in the Use of Water-Power.

BY C. M. MYRICK, SAN FRANCISCO, CAL.

(Cleveland Meeting, October, 1912.)

THE following notes are submitted in the belief that they may interest some of the many owners of small water-power plants, so generally used in mining-work throughout the West.

A small and somewhat primitive quartz-mill was taken over by a leasing company, and a 10-stamp battery was installed. Originally, power had been furnished by a Pelton wheel; but, since the supply of water depended on the rate at which snow was melting on the mountains, a steam-plant had been added to help out during cold spells. This auxiliary plant was antiquated and expensive to operate, and, soon after starting up the remodeled mill, it broke down completely; so that it became necessary to get all the power possible out of the Pelton wheel, with its ever-varying water-supply.

The pipe-line, about a quarter of a mile long, was made up of assorted sizes, from 7 to 11 in. in diameter; and, since it was buried deep under snow, there were no data from which to calculate the theoretical size of the nozzle. The stream being at this time at a low stage, it was important to make the most efficient use of the available water. This was done by using a nozzle large enough to pass the whole supply—taking care, however, to limit the size so as not to lower the water-level in the tank at the head of the pipe-line.

As the weather became warmer, the nozzles were gradually enlarged to meet the increasing volume of water; but a point was finally reached where further enlargement only gave decreased power. Evidently the loss from friction had more than balanced the gain from the increased flow through the pipe. However, the area of nozzle that developed the maximum power from the pipe-line had been found. This was used; and the surplus water was allowed to run to waste at the head of the line.

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All this experimenting necessitated the trial of nozzles of many different sizes; and, since these were not on hand, they were improvised as needed. Fig. 1 shows the quick and easy way in which this was done. A hole, 0.25 in. deep and of just the diameter of the outer rim of a cast-iron nozzle that had been found too large, was bored into a plank. Then, using a bit of the size desired for the new nozzle, the hole was continued through the plank. A tapering wooden plug, 8 or 10 in. long, was made to fit into this hole. The cast nozzle was forced into the shallow cut in the plank, the inner plug put in place, and the space between plug and nozzle filled with bab-

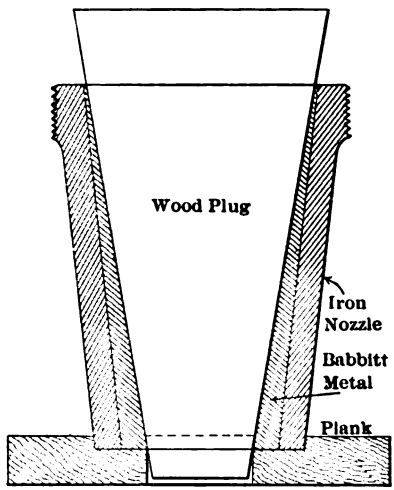

FIG. 1.—METHOD OF REDUCING THE SIZE OF A NOZZLE FOR A PIPE-LINE.

bitt-metal. The wooden plug being removed left a nozzle of the desired diameter. When this particular size was no longer needed, the babbitt bushing was easily removed by heating.

To determine the relative power furnished by the different nozzles, it was only necessary to count the number of drops of the stamps—a method which may seem somewhat crude, but which certainly showed the effective work being done.

In order to get a record of the pressure under the various conditions, the pipe was tapped close to the nozzle, and a steam-gauge, taken from the boiler, was attached. The note-book containing a memorandum of these tests was destroyed

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in the great fire at San Francisco; but the general facts are: that, with the valve closed, the static pressure in the pipe was 60 lb. per sq. in.; that, no matter how much water was available, it was harmful to use a nozzle larger than $2\frac{3}{8}$ in.; and that even a change of $\frac{1}{16}$ in., either way, caused a perceptible slowing-down of the wheel.

Discharging through this $2\frac{3}{8}$-in. nozzle, the pressure stood at 40 lb.; or, in general terms, water from this pipe-line was yielding its maximum power when one-third of the head was lost in overcoming pipe-resistances. Subsequent consultation of authorities showed that this agreed very well with the results obtained from the theoretical treatment of such problems.

Care was taken during the trial to lag up the pulleys, so that the circumferential speed of the water-wheel approximated to one-half the velocity of discharge from the nozzle, calculated from the effective head. It was noted, however, that a small variation was not important.

Incidentally, these experiments demonstrated that there was a very considerable waste of power if the cam-faces were not kept properly greased; and that this loss occurred long before the cams were dry enough to throw sparks. On the other hand, there was no necessity of excessive lubrication, incurring the risk of having grease thrown on the amalgamating-plates.

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A Graphic Solution of D'Arcy's Formula for the Transmission of Compressed Air in Pipes.

BY NATHANIEL HERZ, LEAD, S. D.

(Cleveland Meeting, October, 1912.)

THE formula very frequently used for computing the economical size of pipe to transmit compressed air is that of D'Arcy, as follows:

$$D = c\sqrt{\frac{d^5(p_1 - p_2)}{w_1 l}}$$

Where,

D = the volume of compressed air delivered in cubic feet per minute, at the final pressure.
c = an experimental coefficient determined for the various sizes of pipe.
d = the diameter of the pipe in inches.
l = the length of the pipe in feet.
p_1 = the initial gauge-pressure in pounds per square inch.
p_2 = the final gauge-pressure in pounds per square inch.
w_1 = the density of the air, or the weight in pounds per cubic foot at the initial pressure, p_1.

The most common case is that in which the given quantities are: the quantity of air required, the length of the pipe, and the initial pressure. The method of solution is to assume a pressure-loss and to compute the remaining factor, $c\sqrt{d^5}$, thus giving the size of pipe corresponding to the assumed loss of pressure. It is always desirable to try two or more pressure-drops, in order to find the combination that is most satisfactory, since often a small change in the size of pipe will reduce or increase the loss of pressure several pounds. An alternative method is to assume a size of pipe and calculate the corresponding pressure-drop. Each method involves a series of tedious calculations to arrive at the most economical solution, and also requires the use of tables giving the constant, c, the density of the air, w_1, and often, for convenience, a table giving the values of the expressions $\sqrt{\frac{p_1 - p_2}{w_1}}$ and $c\sqrt{d^5}$.[1]

I have constructed a graphic chart, Fig. 1, for the solution of

[1] W. Cox, *Compressed Air Information.* Edited by W. L. Saunders, p. 297.

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these problems with no computation, and without the use of tables. The procedure is as follows:

Begin with the quantity of compressed air delivered, on the left-hand vertical scale; follow across horizontally to the intersection with the inclined line corresponding to the length of the pipe-line; pass up vertically to the inclined line corresponding to the initial pressure; then across the chart horizontally to the heavy line at the right of the cross-sectioned part of the chart. The point here found is a pivot-point, which is held with a pencil, pen, or needle-point, and a straight edge placed against it and swung across the "Z" diagram. Any two points on the inclined and vertical lines that are cut by the straight-edge at the same time go together as one solution of the problem, giving a pipe-diameter with its corresponding loss of pressure. By swinging the straight-edge, it is possible to see at a glance how the final pressure is affected by a variation of 1 in. in the pipe-size. Moreover, the size giving the most desirable result is determined at one operation. If the drop is considerable, it may be desirable to adjust the volume to correspond with the new final pressure, and to repeat the operation; but, within ordinary economical limits, the error involved by not doing so is negligible.

Sometimes the problem may arise in another form—for instance, to find the maximum volume that can be handled in an existing line. In this case, the process is reversed. Begin with the maximum desirable drop, and the size of pipe, then pass to the initial-pressure line in a horizontal direction, then vertically to the length-line, and finally horizontal to the left-hand scale, which will give the corresponding volume.

Another case is that in which the diameter and length of pipe and the volume of air delivered and the final pressure are given: required, the initial pressure and the pressure-loss. The formula requires the knowledge of w_1, which depends on p_1, unknown. It is possible to substitute the value of w_1 in terms of p_1 and solve,[2] but the usual method is to assume w_1 for a preliminary calculation, and then recalculate, using the new value obtained. This is the method to be used with the chart. It is sufficient, unless the pressure-drop is excessive, to assume that the initial and final pressures are close enough so that the final pressure may be used for the preliminary operation.

[2] W. Cox, *loc. cit.*, p. 301.

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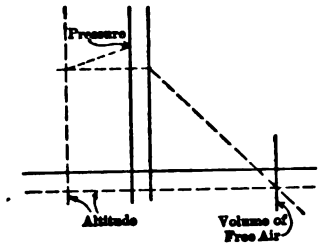
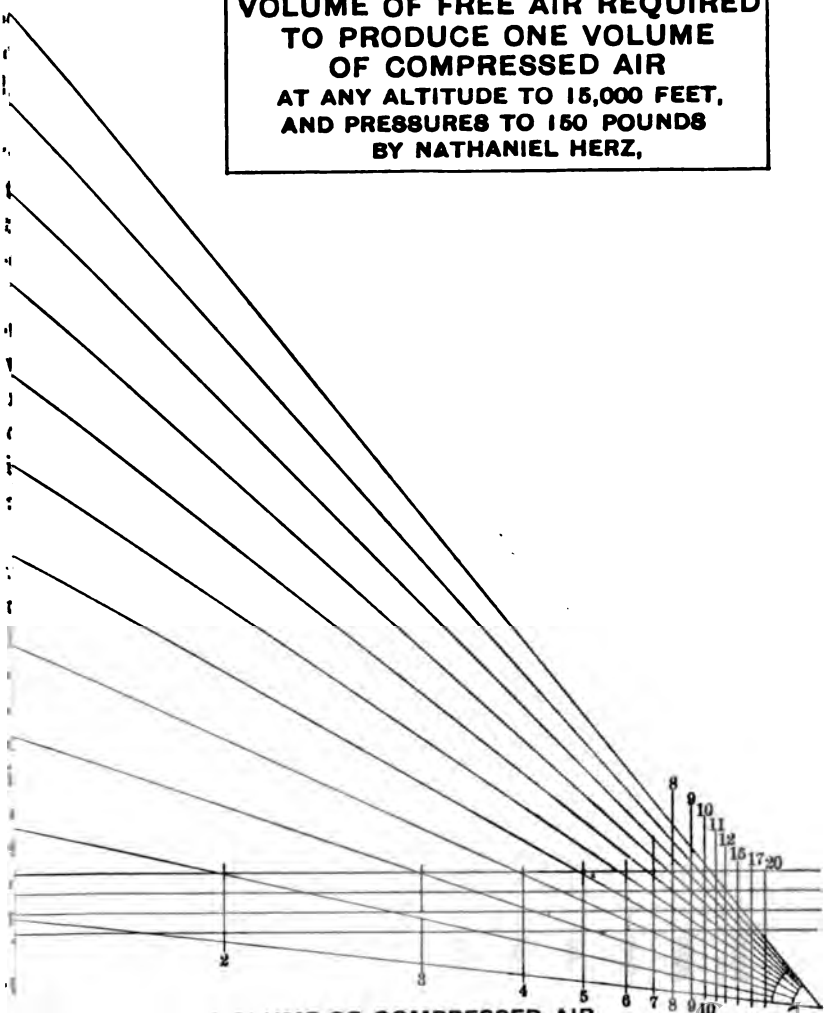


CHART FOR FINDING THE VOLUME OF FREE AIR REQUIRED TO PRODUCE ONE VOLUME OF COMPRESSED AIR AT ANY ALTITUDE TO 15,000 FEET, AND PRESSURES TO 150 POUNDS BY NATHANIEL HERZ,



AIR FOR ONE VOLUME OF COMPRESSED AIR

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Finally, given the initial and final pressures, pipe-diameter, and quantity of air delivered, what is the limiting length of pipe? Start with the pressure-drop, lay a straight-edge across the point corresponding to the pipe-diameter to the heavy line at the right of the square diagram, and run horizontally to the initial-pressure line. Drop a vertical from this point, and find its intersection with a horizontal line drawn through the point on the left-hand scale corresponding to the volume required. These lines will intersect on the required length line.

The accuracy of this chart is well within commercial limits. It has been checked against calculated values for combinations varying from 100 to 1,000 cu. ft. of compressed air delivered per minute, pressure-losses from 3 to 10 lb., and pipes from 100 to 4,000 ft. long; all results were within 0.5 in. of the pipe-diameter, and most of them within 0.25 in. or less.

The constants used were taken from Peele, *Compressed Air Plant for Mines*, p. 197. The pipe-diameters are actual:[3] for standard lap-welded pipe interpolation may be necessary, although in most cases the error involved by not doing so will be negligible.

To add to the usefulness of this chart, I have also worked out a straight-line chart, Fig. 2, for finding the relation between the volume of free air at any altitude up to 15,000 ft., and the volume of compressed air at pressures up to 150 lb. per sq. in. This diagram is the $pv = c$ curve with altitude corrections, thrown into a straight-line chart, all the lines intersecting at the lower right-hand corner. It is used as follows:

Begin with the gauge-pressure on the vertical scale of pressures, using the left-hand line; pass parallel to the slightly inclined lines and intersect the vertical line corresponding to the altitude; then go horizontally to the right-hand vertical pressure-scale. Extend a straight-edge from the point just found to the intersection-point in the lower right-hand corner of the chart, and the point where this crosses the horizontal line corresponding to the altitude gives the volume of free air required to produce one volume of compressed air. This chart is most accurate for the lower pressures, but even for higher pressures it is sufficiently close for most purposes that arise in practical work in compressed-air transmission.

[3] Through an error in the chart as it appeared in *Bulletin* No. 72, December, 1912, the pipe diameters were marked nominal.

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Notes on Bag-Filtration Plants.*

BY A. EILERS,† NEW YORK, N. Y.

(Cleveland Meeting, October, 1912.)

NOTES ON THE BAG-HOUSE.

In Connection with Lead Blast-Furnaces and Leady Copper-Matte Converters.

THE use of the bag-house for filtering out fumes produced in certain metallurgical operations is not new in America. There are no patents in force at this time, to my knowledge, which might hinder such use.

Up to about 25 years ago the bag-house was only employed in connection with either zinc-works, which produced the oxides, to be sold for paint, or as an auxiliary in extracting lead from high-grade oxidized ores and from galena-concentrates in the Missouri and neighboring lead-regions; this work being then done in the Scotch or American hearths and in very low slag-furnaces, in all of which was produced an excessive amount of oxides which were also finally sold as paints. The literature[1] on the above is abundant in text-books and periodical publications.

The first large bag-house to be used permanently in connection with lead blast-furnaces was erected, as far as I am aware, about 1887 at the Globe Smelting Co.'s works at Denver, Colo., under the Lewis-Bartlett patents. The brick flue, running from the blast-furnaces to the bag-house, was only a few

* Presented also at the Joint Meeting of the Institute and Section IIIa, Metallurgy and Mining, of the Eighth International Congress of Applied Chemistry, New York, N. Y., September, 1912, and here published by mutual agreement.

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[1] Ingalls, *Production and Properties of Zinc*, pp. 13 and 14 (1902). Ingalls, *Lead and Zinc in the United States*, pp. 67, 68, 69 (1908). Ingalls, *The Metallurgy of Zinc and Cadmium*, pp. 669, 673 (1903). Hofman, *Metallurgy of Lead*, pp. 131, 144, 145, 393 (ed. 1904).

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hundred feet long. As their furnaces at that time were run with hotter tops than should have been the case, the gases could not be cooled in the flues before reaching the bag-house to a sufficient degree to allow most of the H_2SO_4 to drop out before reaching the bags and to prevent the latter catching fire rather too often. In short, the experiment was not counted an unqualified success among metallurgists and not imitated for many years. About 1902 and even earlier the lead- and other ores in the West had mostly changed to sulphides, so that much roasting was necessary preceding blast-furnace smelting, and in consequence lead-losses became greater than before. At that time and in subsequent years I had many experiments made at various works of the American Smelting & Refining Co., with a view to reducing the temperature of the gases below the danger-point and dropping out most of the H_2SO_4 and As_2O_3 in the flues, before the gases reached the fan and the bags. As the length of the flues in nearly all the works had been much increased before this time with a view to settling flue-dust and part of the fumes by means of Freudenberg plates, or interposed dust-chambers with many partitions, it remained to replace parts of the brick flues in the proper places with sections of iron or of concrete flues for the purpose of increased radiation. It was finally found that in flues of the proper large size and of a length of from 1,000 to 1,800 ft. (depending on the outside average temperature in the locality) there was no difficulty in accomplishing the end sought. It was also found, that with the above length of flues and the devices put in them to give large surface contact for the dust and fumes, very nearly all the silver and gold were precipitated in the flues, hardly any reaching the bags, while a large amount of lead in the shape of fumes traveled on, and would have been lost without a filtering-apparatus or some spraying arrangement, such as is in use in Tarnowitz. The bag-house was considered preferable in America and gradually quite a large number of these were attached to various works.

The object of the present paper is to discuss the employment of the bag-house in connection with lead blast-furnaces and with such copper-converting plants as treat mainly leady copper-mattes, derived from lead blast-furnace smelting.

I have never seen the economical side of such employment

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in print, and it will therefore perhaps be of interest to give some figures in that connection in addition to a brief description of certain establishments.

Out of a considerable number of smelting-plants which employ the bag-house for the collection of fumes, two have been selected for illustration, each of which is representative of its class.

One of these treats the ordinary ores of the West, as they come into the market and are bought by the custom smelters. The furnace-charges made from these ores are low in lead (from 10 to 12 per cent.) and in the precious metals. They also contain much moisture and up to 4 per cent. of sulphur.

The other treats the slagged oxides coming from a very large lead-refinery, leady copper-mattes for concentration, and some ores necessary for slag-making. The furnace-charges in this case are high in lead, often 40 per cent. and over, and rather low in the precious metals. The same plant also converts concentrated, leady, copper-matte to blister-copper. A separate bag-house is provided for this operation.

I am indebted to A. H. Richards, Assistant Superintendent of the Murray plant of the American Smelting & Refining Co., for a detailed description of its bag-house and its operation, together with a number of drawings, all of which is given here with very few changes.

The Murray Plant Bag-House.

General Construction of Bag-House and Accessories.

The Murray plant bag-house was constructed in the year 1906–1907, and was put into actual use July 7, 1907. The total cost for construction was $127,194.89 and included in this amount was cost of 4,032 cotton bags, of distributing-flue the entire length of bag-house, one 15 by 6 ft. Buffalo Forge fan, one 100-h-p. motor, fan-house, small bath-house, and about 100 ft. of 17 by 16 ft. brick flue to connect with blast-furnace flue. An elevation and sections of the bag-house are given in Fig. 1 (Drawing No. 1798), and the fan-house and machinery are shown in Fig. 2 (Drawing No. 1853).

Construction and Size of Bag-House Building.

The outside dimensions of bag-house are, 216 ft. 6 in. by 90 ft. 6 in., height to roof-trusses being 51 ft. 6 in. Construc-

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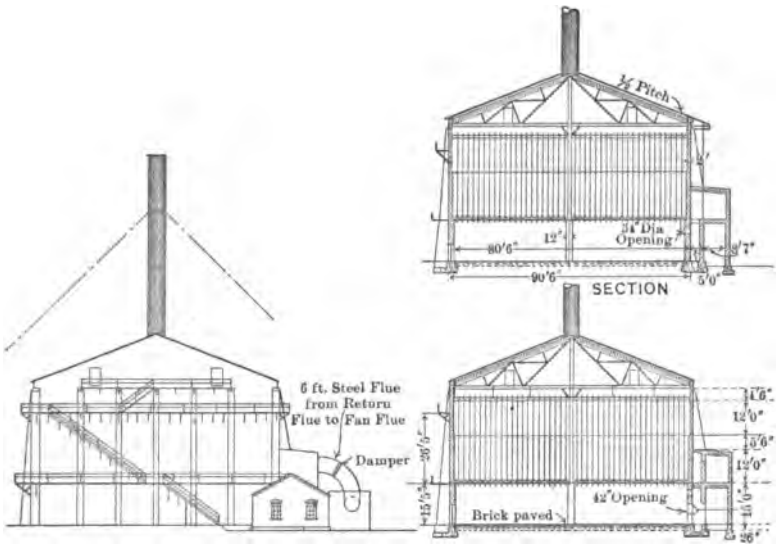
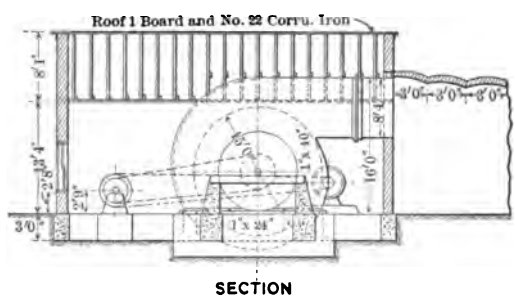
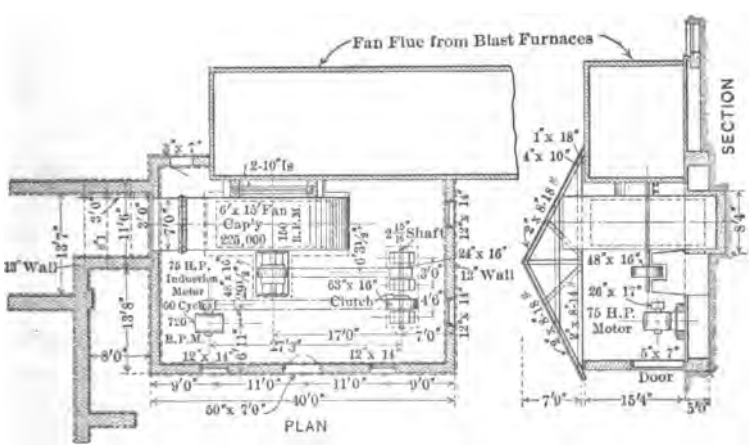


(Works drawing No. 1798.)

FIG. 1.—REVISED LATERAL SECTIONS AND ELEVATIONS OF BAG-HOUSE.





(Works drawing No. 1853.)

FIG. 2.—PLAN AND MACHINERY FOR FAN-HOUSE.

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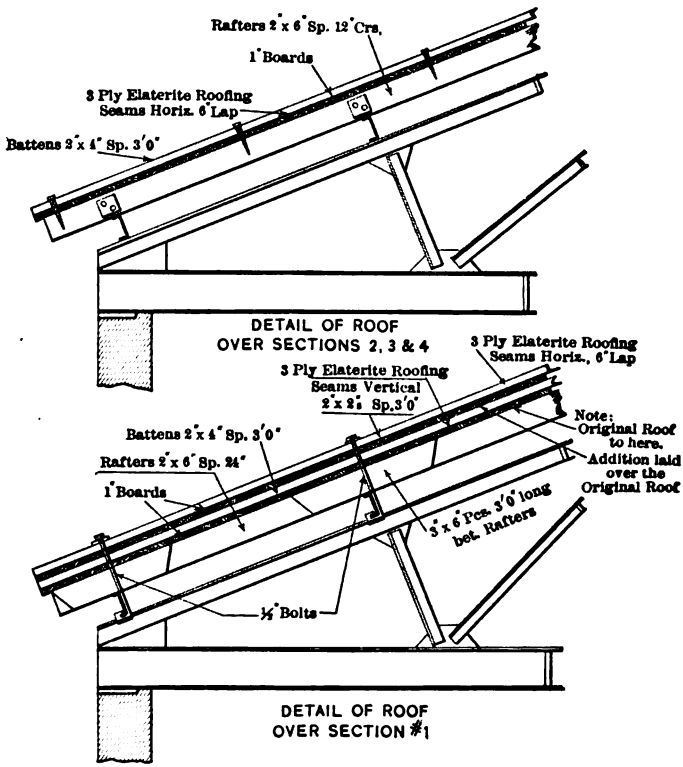
tion is of brick, outside walls being 21 in. thick for a height of 16 ft., then reduced to 17 in. for 18 ft., and to 13 in. for the remaining 17 ft. 6 in. Twenty-four-inch brick buttresses projecting 4 ft. at floor-line and tapering to about 1 ft. at roof-truss line, are built into the walls to stiffen them. Fifteen buttresses are built on each side of the building, and five on each end.



(Works drawing No. 2075.)

FIG. 3.—SKETCH SHOWING CONSTRUCTION OF BAG-HOUSE ROOF, AMERICAN SMELTING & REFINING CO., MURRAY, UTAH.

Roof-Construction.

Fig. 3 (Drawing No. 2075) shows in detail the evolution in the construction of the bag-house roof. Originally the roof consisted of 2- by 6-in. rafters spaced 3 ft. apart and fastened to 6-in. channels by means of an angle-clip which was bolted to the rafters and to the channels, 1-in. Oregon fir common boards nailed to the rafters and 3-ply elaterite roofing nailed on these boards with vertical seams, the seams being laid in tar-paint. This roof

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commenced to leak after a few months, due to the corrosion of the brads holding the elaterite roofing in place. These brads were eaten to such an extent that when a compartment was open and a slight wind was blowing, the elaterite roofing would be lifted off from the boards in places. Considerable patching was done during the first two or three years the bag-house was in operation. After two years the nails holding the 1-in. board sheeting were eaten to such an extent that a heavy wind blew about one-third of the roof off from one compartment. The sheeting itself showed dry rot, but would have lasted some time if the nails had not been partly eaten up. It was evident that nails could not be depended upon for holding the roof down. In the early part of 1910 a new roof was put on top of the old roof in No. 1 compartment, allowing an air-space between the two roofs. Two- by 4-in. battens, 3-ft. centers, were laid on top of the old roof, 1-in. sheeting nailed on top of this, elaterite roofing with horizontal seams nailed to this, allowing a lap of about 4 in.; 2- by 2-in. battens spaced 3-ft. centers were put on top of the elaterite roofing and the two roofs were bolted to the 6-in. channel purlins. To give added strength, 3- by 6-in. pieces 3 ft. long were put on the 6-in. channels, between rafters, the 2- by 4-in. and 2- by 2-in. battens coming directly on top of these and bolts going down through them. This roof has been in use for over two years, and is in excellent condition. The air-space appears to prevent any condensation and keeps the bags drier than in other compartments. As this new roof over the old one was rather an experiment and the other three compartments were badly in need of a roof, new roofs were put on them similar to the upper sketch in Fig. 3. The old rafters were in fair condition, but additional rafters were put in, making the rafters 1-ft. centers; 1-in. sheeting was nailed on top, then elaterite roofing with seams horizontal, then 2- by 4-in. battens 3 ft. apart. Lag-screws were used to hold 2- by 4-in. battens to the rafters, making a roof that will stand as long as the rafters are good, or until the elaterite roofing disintegrates. To prevent the 2- by 4-in. battens from cutting into the elaterite roofing washers of elaterite were used under all of them. The roof of these three compartments has been in use a trifle under two years, but has never leaked and is in good condition.

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Exhausting Gases from the Bag-House.

Each compartment has a steel stack coming out from the apex of the roof and about the center of the compartment. The four stacks are each 6 ft. in diameter and 78 ft. high. The general arrangement of these stacks on the bag-house can be found in Fig. 4 (Drawing No. 1869). When the stacks were built the joints were lapped in such manner that when there is condensation some of the moisture finds its way to the outside

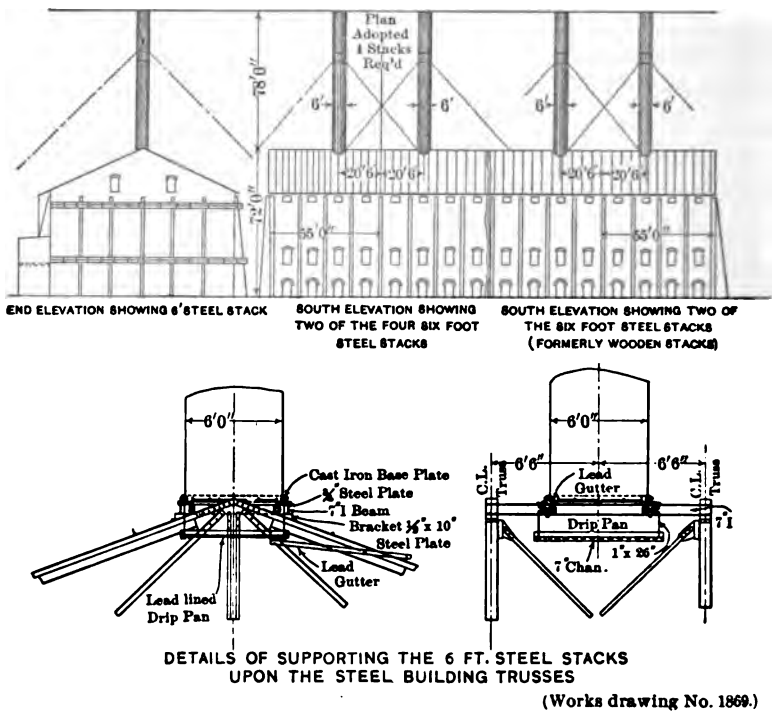

FIG. 4.—DETAILS OF SUPPORTING THE 6-FT. STEEL STACKS UPON THE STEEL BUILDING-TRUSSES.

of the stacks, continually giving them an unsightly appearance. This also weakens the stacks, due to the corroding influence of the weak acid solution going through the joints. The four stacks were found to be corroded to a considerable extent and one or two holes can be observed at a distance of about 32 ft. from the top. About every two years the stacks have been painted on the outside.

To take care of the condensed moisture that trickles down

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the inside of the stack, a lead-lined pan is hung under the stack at a sufficient distance so as not to interfere with the draft. From this lead-lined pan a lead gutter runs out on to the roof. At certain times of the year considerable solution is caught in these pans and runs on to the roof. We had some very sad experiences by trying to use an iron elbow and iron pipes to lead this solution to the roof, as the iron pipe would eat through and before we knew of the condition several bags would be ruined by the weak acid falling on them. The lead gutter with the lead connection from the pan solved this difficulty and gives very good satisfaction. In the lower part of Fig. 4 is shown the lead-lined pan and gutter under the stacks.

Compartments and Cellars.

The bag-house is completely divided into four compartments by means of brick partitions running from the floor-line to the apex of the roof. These partitions are the same thickness as the outside walls. Below the thimble-floor, each compartment is divided into four chambers or cellars by means of 13-in. brick partitions. These cellars are 87 by 12 ft., each one having a 4- by 8-ft. iron door through which the dust can be removed. For plans of the different floors see Fig. 5 (Drawing No. 1756).

Distributing-Flue.

A 16- by 16-ft. distributing-flue built of brick, jack-arch roof construction, runs the entire length of the bag-house and connects with each of the 16 cellars by means of 42-in. bullseye "valves." The dampers or gates for these valves are made of 0.25-in. sheet steel stiffened by angle-irons. These dampers are hinged at the top and open into the distributing-flue. They are operated by means of 2-in. gas-pipe stems running through the distributing-flue.

Return-Flue.

On top of this distributing-flue is another smaller flue, 11- by 16-ft., running the entire length of the bag-house and connected to each cellar by means of 34-in. bullseye valves. The dampers in these valves open into the cellars and were originally operated by means of gas-pipe stems running through the flue, but later, due to the stems corroding and not operating with ease, they were abandoned and the dampers are now closed and

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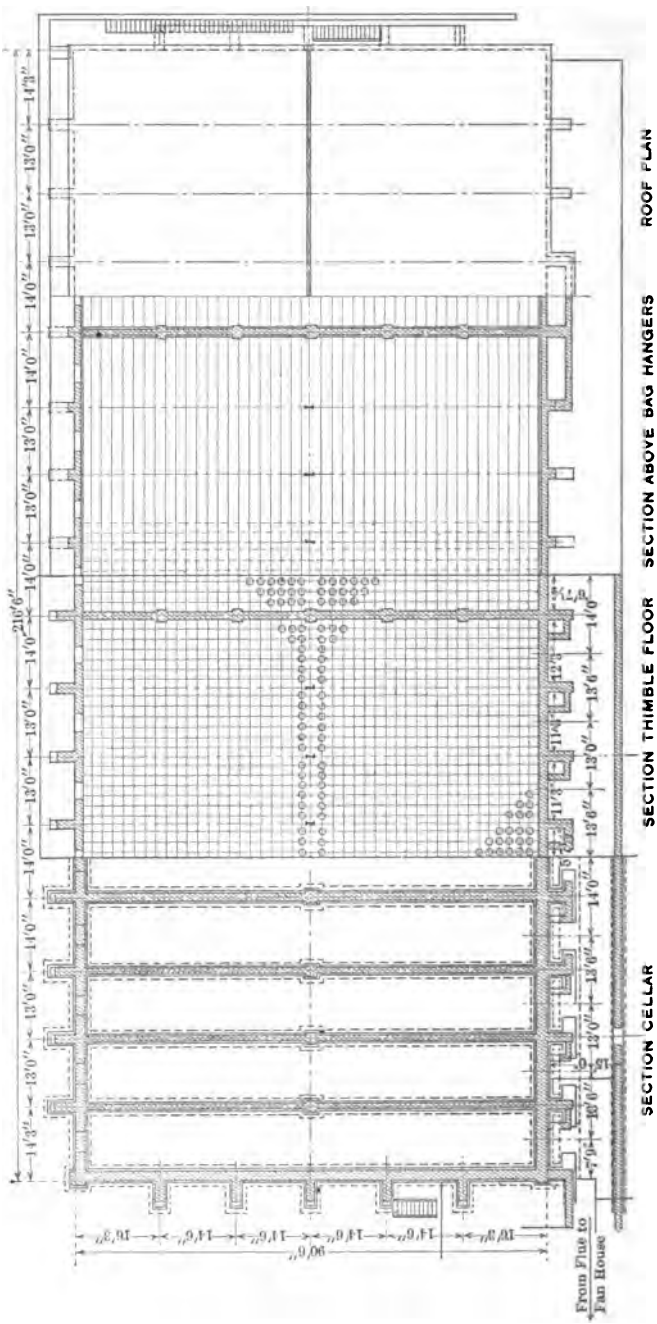

FIG. 5.—REVISED PLANS OF BAG-HOUSE.
(Works drawing No. 1756.)

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opened by means of a chain on the thimble-floor. This upper flue is connected to the bag-house fan-flue by means of a 6-ft. sheet-iron downtake containing a butterfly-damper, and is used to conduct the gases from any chambers that are in the sintering process, back to the fan and thence into the distributing-flue, where they are distributed to the cellars in which the fume is accumulated.

Thimble-Floor.

The bag-house thimble-floor, as has been previously stated, is divided into four compartments, each compartment containing 1,008 steel thimbles 17 in. in inside diameter and 10 in. high. These thimbles are made of No. 14 steel plates and are riveted to the steel floor, which is made of No. 10 steel plates.

Hanging and Shaking of Bags.

The bags are 18 in. in diameter and 30 ft. long. The original 4,032 cotton bags were wired at the top and suspended by means of this wire from hooks fastened to small 3-in. channel-irons. To shake the bags it was necessary to "air out" the compartment completely, then go in between two rows of bags, shaking the bags on each side by hand. In 1910 each row of bags was hung on a 1.5-in. gas-pipe which was suspended by a "U" bolt from the old bag-hooks. By means of a lever on the outside of the building, this gas-pipe could readily be given a horizontal motion of about 5 in., the pipe coming to a sudden stop by means of a collar striking an iron plate on the inside of the bag-house wall. In addition to the horizontal motion, about 0.5 in. vertical motion is imparted, which is rather beneficial in removing the dust from the bags. Forty-two bags can thus be shaken by means of one lever; this requires not more than 3 or 4 min. This shaking-device has proved extremely effective at the Murray plant and has many advantages, I believe, over the system used at the Selby plant, where gas-pipes are run on ball-rollers. The ball-runs and ball-rollers have a tendency to accumulate moisture and fume, making it necessary to clean them at intervals. The construction of ball-runs and rollers is more expensive than that of the "U" bolt-hangers; also no vertical motion is imparted when the gas-pipe is run on ball-rollers. When this shaking-device was installed it was decided to have all bags folded over for 3 in. and sewed

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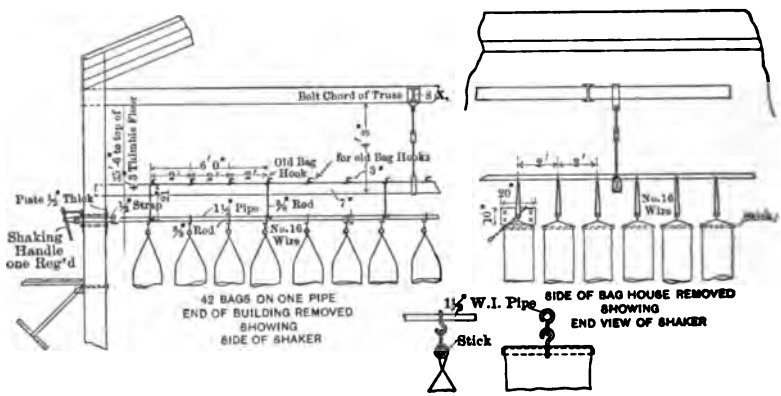
across at the lower part of the fold. This allowed us to put a broom-handle or any piece of round stick through the bag and hang up by means of this stick. The old method of gathering the top of the bag and fastening it with wires has a tendency to allow the dust to stick in the folds and in time to rot the bags at the top. With the stick running through the fold, the bags stay open in better shape and also there is no wire to come in contact with the fiber of bags. Fig. 6 (Drawing No. 1901) shows the general design of the bag-shaking device. Our method for hanging bags to the shaker by means of "S" hooks is given in the lower and middle portion of this drawing; no wire is used at all. We find this means of hanging bags very simple and effective.



(Works drawing No. 1901.)

FIG. 6.—BAG-SHAKING DEVICE FOR AMERICAN SMELTING & REFINING CO., MURRAY, UTAH.

Bags, Cost, Etc.

Four thousand and thirty-two bags 18 in. in diameter by 30 ft. long were originally installed and put into use July 7, 1907. These cotton bags cost $2.136 apiece, f.o.b. Murray, and were purchased from the Denver Dry Goods Co., Denver, Colo. At the end of two years it was computed that the average life of the cotton bags was 17 months, 11 days. Owing to no drip-pans being provided under the four steel stacks, many bags were lost in the first eight months, due to condensation of moisture in these stacks and the consequent dripping of this moisture (containing small traces of acid) on to the bags. In fact, during the first eight months 588 bags were replaced. The leaky roof after about one

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year's run was also instrumental in destroying many bags. Had it not been for these two items of weak construction, there is no doubt but what the life of cotton bags would have been longer than 17 months, 11 days. The rotting of these cotton bags generally commences near the thimble and to a less extent near the top.

In May, 1909, No. 4 compartment was equipped with woolen bags purchased from Buell Manufacturing Co., St. Joseph, Mo. These bags cost $4.7185 apiece, f.o.b. Murray. The compartment was put into commission May 19, 1909, and to date there have been no bags replaced. About 12 bags have been lowered 1 ft. during this time, due to rotting near the bottom. The fiber of the bags is quite strong as yet, and I would estimate that the average life will be somewhere around four years. Shortly after these bags were put into commission about one-third of the roof on No. 4 was blown off, and a rain storm occurring at the same time, rather drenched at least half of these woolen bags, with no apparent ill-effect. The woolen bags in this compartment are all hung by means of wire twisted around the tops of the bags. At the present time we have no cotton bags in service, and are completely equipped with woolen bags with the exception of 273 thimbles that are not now in use, but will have woolen bags put on them shortly. These 273 thimbles were reserved to use up the old cotton bags, the last of which were rotted away this last winter.

Below is a tabulation of the bags used to date:

	Bags.	Cotton.	Woolen.
Original installation July 7, 1907	4,032	4,032	.....
Year ending May 1, 1908	752	752	.....
Year ending May 1, 1909	3,107	3,107	.....
Year ending May 1, 1910	4,073	2,088	1,985
Year ending May 1, 1911	1,493	.....	1,493
10 months ending March 1, 1912	281	.....	281
Total	13,738	9,979	3,759
On hand in bag-house	3,759	none	3,759
Total destroyed	9,979	9,979	.....

I might add that 90 woolen bags were purchased from the Laporte Woolen Mills, Laporte, Ind., at an average cost, f.o.b.

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Murray, of $4.784. These bags were installed about two years ago and appear to be of a good quality.

Five hundred and seventy thousand and twelve sq. ft. of filtering-surface was provided for filtering 165,000 cu. ft. of gas per min., or at the rate of 3.45 sq. ft. of filtering-surface per cu. ft. of gas per min. When one compartment was shut down for sintering, however, there would be 2.59 sq. ft. of filtering-surface per cu. ft. of gas per min., and as it was expected that one compartment would be down most of the time, 2.5 sq. ft. was the basis used for calculating the number of bags needed.

Fan.

A Buffalo Forge exhaust-fan with impeller 6 by 15 ft. brings the gases from the blast-furnaces and forces them into the distributing-flue and through the bags. This fan is capable of handling 1,500 cu. ft. of gas per rev. at a pressure of 1 in. of water. A 100-h-p. motor was formerly used when the fan was running at a speed of 93 rev. and handling gases from 8 blast-furnaces. Sixty i.h-p. was required. Later a 50-h-p. motor was installed as we had only 5 and 6 furnaces in blast. Indicated horse-power required when running at 80 rev. per min. with the 50-h-p. motor is about 45. The countershaft is equipped with ring oil-bearings and inner fan-bearings are water-jacketed. Very little difficulty has been experienced with this fan since its installation. Pressure in the distributing-flue is usually about 0.5 in. of water, at times going up to 0.75 in. just before shaking the bags.

Operations.

Conducting Gases from Furnaces.—The blast-furnace gases are conducted through 795 ft. of brick flue and 70 ft. of sheet-iron goose-neck. Of this brick flue, 725 ft. is 16 by 17 ft., walls 17 in. thick, roof 8-in. jack arches, 5-ft. centers. Seventy ft. of the brick flue is 16 by 20 ft., walls 17 in. thick, roof of reinforced concrete slabs 4 in. thick. The sheet-iron goose-neck is 20 ft. in diameter, made of 0.25-in. steel. Back of the blast-furnaces is 262 ft. of 16- by 20-ft. flue with sloping bottom, so that No. 1 blast-furnace is about 865 ft. from the bag-house fan, while No. 8 blast furnace is about 1,127 ft. from the bag-house.

The temperature of the gases leaving the blast-furnaces varies somewhat. Certain readings showed 119° F. in flue near No. 1

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blast-furnace and 85° F. at bag-house, showing a drop of 34° F. in the 865 ft. of flue when outside atmosphere was 64° F. No tests have ever been made at this plant to find the SO_3 and arsenic that were thrown out in the flues due to the drop of temperature. The gases entering the bag-house were analyzed, however, at a time when there was quite a deterioration of the bags and were found to contain 0.003 per cent. of SO_3. During the winter months the temperature of gases entering the bag-house fan is often quite low. The tabulated statement below will show the variation of temperature due to the season of the year.

	1909.	1910.	1911.
January	81.4° F.	83.3° F.	79.9° F.
February	73.0	93.0	79.9
March	83.9	113.4	83.47
April	91.3	107.2	93.4
May	102.0	106.3	108.65
June	115.5	124.2	114.5
July	117.1	131.0	123.9
August	115.3	130.9	120.5
September	99.46	110.7	114.3
October	105.68	93.7	93.9
November	103.02	89.15	90.7
December	94.6	87.0	81.9
Average	98.52	105.82	98.75

Details of Bag-House Operation.—Gases are discharged from the fan into the distributing-flue; dampers in lower bullseye valves being open the gases enter the 16 cellars, filter through the 4,032 bags; filtered gases discharging through the four steel stacks on top of the bag-house. Once every 24 hr. all the bags are shaken, the operator using the outside lever and shaking each row of 42 bags. All doors in compartments are kept closed while shaking bags so as not to allow the temperature in the bag-house to go down too low. The shaking-operation is done by first closing the four dampers in the lower valves of one compartment, shaking the bags, opening the valves again and taking the next compartment in turn until the bags in all four compartments have been shaken. This usually occupies the attention of one man for about 3 hr. each day and could be done much quicker if necessary.

On the first of each month one cellar door in each compart-

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ment is opened, the dust measured and representative sample taken. This is done for the purpose of taking stock. The doors are then closed and sealed by means of mortar. When the dust has accumulated to a thickness of about 24 in. the entire compartment is closed off from the distributing-flue, cellar doors are opened, valves connected with the return-flue are opened and fume in the four cellars is lighted by means of hot coals. Formerly we set fire to the fume across the front, allowing the burning to creep gradually to the back end, but as this generally took from 10 to 14 days to complete the sintering process, we changed the method and now throw the hot coals all over the surface of the fume. This apparently gives a much better sinter and the operation is complete within 5 or 6 days so that the fume can be removed. Spontaneous ignition was rather frequent the first two or three years we were operating the bag-house, but of late it is rather rare. The exact cause of this spontaneous ignition has never been definitely determined at this plant, but we have found that it generally occurs when at least 24 in. of fume has accumulated and we believe it to be due to some development of heat when a large mass of the fume has accumulated. Of late we do not allow the fume to accumulate over 24 in. and spontaneous ignition has been very rare.

The sintered fume is removed from the cellars by means of laborers using wheel-barrows and shovels, and is loaded into standard-gauge box-cars and shipped to the Globe plant at Denver. While removing the fume from the cellars every tenth wheel-barrow is reserved as a sample to determine the values of the lot, this sample being carefully coned, quartered, and the reject loaded into cars with the main portion of the fume. Except at times when sintered fume is being removed from the cellars, one man on day-shift attends to the operating of the bag-house. For the 16 hr. he is away from the plant our H. & H. oilers come over to the bag-house once every 2 hr., take a temperature-reading and look after the oiling of the bearings. These temperatures are taken just at the inlet of the fan. All laborers are required to take a shower-bath and change their clothes before leaving the bath-house. The men removing sintered fume have very little trouble with sore legs and faces, providing they are careful to clean thoroughly after their day's work.

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When the bag-house was first started, rubber suits were furnished to the bag-shaker and foreman, also helmets, but later it was found that these were not essential to the men, and that the main secret of good health of the laborers was to bathe properly and not to eat their lunches with dirty hands and faces. When the bags were shaken by hand, and it was necessary for a man to go into the compartment, between the rows of bags, we had one man shake bags for two years with no ill effect. He realized the necessity of thorough cleanliness.

Assay and Analysis, also Recovery of Fume for 8 Months Ending Dec. 31, 1911.

Charge smelted in 8 months, tons	254,794
Fume produced in 8 months, tons	1,124
Charge recovered as fume, per cent	0.44114
Fume recovered per ton of charge, pounds	8.8228

	Oz. Au.	Oz. Ag.	Lb. Pb.	Lb. Cu.
Total metals in charge smelted	17,319.854	3,472,681.90	50,079,080	1,834,355
Total metals in 1,124 tons fume recovered	16.754	2,948.35	638,861	1,295
Per cent. of metal in charge recovered in fume	0.097	0.085	1.276	0.071
	Oz. P. Ton.	Oz. P. Ton.	Per Ct. Pb.	Per Ct. Cu.
Assay of the 254,794 tons of charge	0.068	13.629	9.827	0.86

	Oz. Au.	Oz. Ag.	P. Ct. Pb.	P. Ct. Cu.	P. Ct. SiO_2	P. Ct. Zn.	P. Ct. S.	P. Ct. As.	P. Ct. Fe.
Assay and analysis of the 1,124 tons unburned fume recovered	0.0149	2.62	28.42	0.058	2.9	3.2	5.2	36.6	5.1
Assay and analysis of sintered fume	0.014	2.4	27.3	0.05	2.5	4.2	4.2	33.5	5.2

Recovery of Fume, Etc., for Entire Operation of Murray Plant Bag-House, July 7, 1907, to Feb. 29, 1912.

	Charge Smelted.	Fume Recovered.	Assay Per Ton Fume Recovered.	Per Cent. of Charge-Contents Smelted.	Value of Metals Recovered.
Weight	1,666,857 tons.	13,857,645 lb.	.....	0.415	.....
Gold	256,874.986 oz.	125.595 oz.	0.018 oz. p. ton.	0.049	$2,587.10
Silver	24,725,274.19 oz.	14,885.62 oz.	2.15 oz. p. ton.	0.063	8,028.25
Lead	354,693,161 lb.	4,492,288 lb.	32.4 per cent.	1.269	141,745.02
Copper	16,521,715 lb.	1,947 lb.	.....	0.0118	330.15
Total	.....	.....	.....	.....	$152,690.52

Operating-Expense.		Outcome.	
Labor	$16,440.05	Treatment-charge 6,929 tons at $10.00	$69,290.00
Motive power	13,542.10	Operating-cost	76,853.43
Supplies and Repairs	46,871.28	Total cost	$146,143.43
Total	$76,853.43	Value of metals recovered	152,690.52
		Gain for 4 years and 8 months	$6,547.09

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The following additional figures may be of interest:

Cost per ton of original charge smelted for operating bag-house	$0.0461
Gain per ton of charge smelted in operating bag-house	0.00393
Fume recovered per ton of charge smelted, pounds	8.31
Average cost of operating bag-house per month	$1,372.38
Average gain in operating bag-house per month	116.91
Total cost for replacing bags in 56 months	$30,453.6285
Average cost per month for replacing bags during 56 months	$543.8148
Cost per ton of charge smelted for replacing bags	$0.0186
Total bags replaced in 56 months	9,979
Total bags replaced per month	178.2

	Au.	Ag.	Pb.	Cu.
Average assay of unburned fume for 56 months	0.018	2.1	32.4	0.014

The above data show the apparent gain from the installation of the bag-house in 4 years and 8 months. But the figure does not include interest on the original investment, nor a fair percentage of same for amortization.

If these two items were included, as they should be, we would have:

Gain in 4 years and 8 months		$6,547.09
6 per cent. interest on cost of $127,194.89 for 4 years and 8 months	$35,614.57	
5 per cent. amortization for 4 years and 8 months	29,678.80	65,293.37
Net loss in 4 years and 8 months		$58,746.28

On account of the advisability of extracting the high percentage of As_2O_3 from the fumes, the Murray fume has to be shipped to another works of the American Smelting & Refining Co., which is equipped with a plant for that purpose and works up fumes from a number of lead-smelting plants. It is not good financial policy to provide every plant with an arsenic establishment, for it would be idle three-quarters of the time.

But if the Murray plant had happened to be the one provided with its own arsenic establishment, the cost would not have been more than $5 per ton, thus diminishing the loss shown above by $34,654, leaving the loss for 4 years and 8 months, still $24,101.28.

The above figures show that, taking the immediate financial outcome only, bag-houses are not profitable in lead-smelting works, which treat ores low in the precious metals, and especially in lead. But there is another consideration to be taken

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into account, when smelting-works are, like Murray, located in the midst of farms and villages. This is the elimination of constant law-suits, which are bound to be of frequent occurrence when As_2O_3 and the sulphates of lead and other metals are permitted to drop down into the fields. It must be decided, therefore, in the case of each smelting-plant, whether it is not good business to expend the money for a bag-house and for operating it, though an immediate profit is not apparent.

W. T. Page, Manager of the Omaha plant of the American Smelting & Refining Co., has kindly furnished me with data on the bag-houses of that plant, which appear below with slight changes and some explanations. The figures given are reproduced as originally sent me.

*Bag-Houses of the Omaha Plant.**

A distinctive feature of the Omaha plant is said to be its fume-condensing facilities. The lead-loss of the plant does not exceed 1 per cent. though none of the gases of the lead-refinery reverberatories enter the bag-house. There are three systems of fume-condensing, as follows:

1. Converter bag-house system.
2. Blast-furnace bag-house system.
3. Zinc oxide bag-house system.

No. 1. Converter Bag-House System.

The gases from two stands of acid-lined converters (upright) and one basic-lined converter (Peirce-Smith type) lead into a suspended flue of cross-section as shown in Fig. 7. It is made of sheet steel and heavily painted inside and outside with a graphite paint. Its length is 1,250 ft. This flue leads to a fan 12 ft. in diameter and 6 ft. wide, which runs at 140 rev. per min. and requires 25 h-p. The fan discharges into a bag-house containing 940 bags, divided into two compartments. The cellar is on the ground level so that carts can get in to load out the fume. It is 10 ft. high from floor to thimble

* No drawings of these bag-houses are published, as the construction is in general the same as the Murray plant. The one difference is in the shaking-arrangement of the bags, which is here done by beating the bags with long slender wooden rods, suspended from a shaft for each row of bags. This shaft can be turned to and fro by means of a lever outside of the building, thus giving the rods a beating motion.

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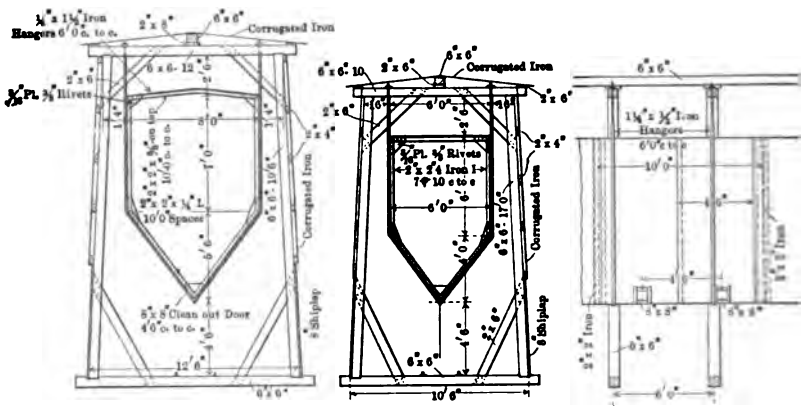
level. Many have advocated a cellar with a sloping floor and a conveyor of some design to remove the fume, but every practical design submitted to date represents too large an expenditure for the advantage in labor saving obtained.

Sometimes it is found necessary to clean out the house three times during the life of one set of bags. The water put on the fume (which is nearly white and like flour) to make it possible for the men to handle it, necessarily allows the bags to absorb moisture to an excessive degree. This will shorten the life of the bags.



BLAST FURNACE FLUE
1400 Feet of Flue
14 Feet Dia. Fan
5'-11" Feet Dia. Fan Inlet
140 R.P.M.
55 H.P. Motor

COPPER CONVERTER FLUE
1250 Feet Long
12 Feet Dia. Fan x 6'
140 R.P.M.
25 H.P. Motor

FIG. 7.—SECTIONS OF FLUES.

With the acid-lined converters one set of bags, on an average, will filter 1,600 tons of lead sulphate fume, representing 4,500 tons of blister-copper.

The secret of long life to bags seems to be the use of long flues conducting the gases to the bag-house and constructed of a material which permits rapid radiation, also a good vacuum above the bags, so that the gases are drawn away rapidly. With this in view, Omaha has connected the converter bag-house to a 300-ft., self-supporting, brick-lined, steel stack.

Average pressure and vacuum are as follows:

Vacuum at fan, up to 0.8 in. (water).

Vacuum at converter, from 0.07 to 0.15 in.

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In the bag-chamber there is a variation in pressure of 0.025 in. to a vacuum of 0.07 in. At the base of the stack, with no other gases entering, there is a vacuum of 0.3 in. With a small reverberatory furnace leading into the stack to supply it with heat in cool weather, there is a vacuum at the base of 0.7 in., while the figures on the bag-chamber remain the same as given.

The pressure in the cellar of the bag-house goes up to 0.56 in. With a vacuum of 0.3 in. at the fan a vacuum of 0.1 in. is found at the converter, with a very regular increase along the flue towards the fan.

Following are some data concerning the converter bag-house:

Size: 102 by 60.8 by 40 ft. outside.

Walls: 17 in. with two compartments up to the thimble-floor; 13 in. above thimble-floor.

Thimbles: 18 in. in diameter, placed 2 ft. 4 in. centers.

Bags: 28 ft. long, 940 in number.

Cubic feet in flue, 68,000.

Cubic feet in cellar, 67,000.

Cubic feet in bag-chamber, 174,000.

Sq. ft. filtering-area of bags, 124,000.

Production about 45 tons of blister-copper a day, from matte assaying Cu, 45; Pb, 23; S, 18 per cent.

The blowing-engine at converters runs about 15 hr. out of 24, or 52,000 rev. per day, with delivery of 100 cu. ft. of air per rev. Pressure, 17 lb. per sq. inch.

An example of flue-temperatures, etc., follows. These vary, of course, with the temperature of the outside air.

Flue-Temperatures (F.) on First or Slag Blow, Outside Air at 80° F.

Back of converter	700
80 ft. from converter	557
200 ft. from converter	464
400 ft. from converter	390
600 ft. from converter	345
800 ft. from converter	300
1,100 ft. from converter	266
At fan	246
At bag-house	152
At stack	136

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Fume-Analysis.

	Ag.	Pb.	Zn.	Combined SO_3.	Free SO_2.			
	Ounces. Per Ton.	Per Cent.	Per Cent.	Per Cent.	Per Cent.			
200 ft. from converter	4.2	56.0	1.6	25.13	0.25			
400 ft. from converter	3.0	60.0	1.6	23.13	0.55			
600 ft. from converter	2.4	64.2	1.3	22.82	0.15			
800 ft. from converter	2.0	64.6	1.3	22.60	0.00			
1,100 ft. from converter	2.1	63.0	1.5	23.33	0.00			
1,200 ft. from converter	1.6	64.4	1.6	20.60	0.00			
					As.	Sb.	Te.	Se.
					Per Cent.	Per Cent.	Per Cent.	Per Cent.
Caught in bag-house....	2.8	66.0	1.9	21.63	2.0	0.2	0.2	0.24

SO_3 Determinations in Gases.

At Converter.	At Fan.
Per Cent.	Per Cent.
0.038	0.020
0.027	0.025

Simultaneous SO_2 Determinations.

At Converters.	At Fan.	At Middle of Flue.
Per Cent.	Per Cent.	Per Cent.
3.62	2.055	1.70
2.34	1.21	1.82
4.46	2.68	.....
2.38	1.45	.....

SO_2 in Bag-Chamber.

Top.	Bottom.
Per Cent.	Per Cent.
2.06	2.06
1.30	1.60
1.52	1.40

Other temperatures taken with corresponding outside air-temperatures are as follows:

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Temperature at Fan.	Air-Temperature.
Degrees F.	Degrees F.
225	45
192	59
239	61

and at another time:

Temperature 60 Ft. From Converter.

	Degrees F.
Slag blow	525
Slag blow	520
Copper blow	625
Slag blow	675
Slag blow	675
Copper blow	600

The average temperature in the bag-chamber is 140° F. with a variation from about 100° to 180.°

The material used in bags is Osnaburg sheeting put up in rolls of about 600 yd., 57 in. wide, at 15 cents per yd. (It costs about 15 cents per bag to cut and sew the material.) The width allows the bag to be made 56.5 in. in circumference, or just right to fit an 18-in. thimble.

Bags are made up 28 ft. long, with a loop at the closed end through which passes a 1-in. pipe or stick to support the bag.

The beater arrangement for the bags is as previously mentioned in a foot-note. The fan is always shut down while beating the bags. This is done about twice a day, or as often as necessary.

Fume Produced from No. 1 Bag-House.

Calendar Year.	Flues.	Bag-House.	Total.
	Tons.	Tons.	Tons.
1906	1,531	1,670	3,201
1907	1,045	3,187	4,232
1908	900	3,333	4,244
1909	568	1,207	1,775
1910	619	2,532	3,151
1911	546	2,703	3,249
	5,220	14,632	19,852
To May 1, 1912	350	725	1,075

As giving some measure of fume collected per ton of blister-copper produced the following may serve:

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Recovered Feb., 1911, Fume, 454 tons.
Fan 152 tons.

606 tons — 15 per cent. water = 515 tons.
100 tons from flue 100 tons.

1667 tons blister produced..... .....615 tons fume.
1 ton blister produced..... ..... 0.368 ton fume.
1 ton blister produced..... ..... 0.221 ton lead.

	Costs.	Blister.	Cost per Ton of Blister.
		Tons.	
1905–6.....	$3,995.14	7,166	$0.558
1906–7.....	6,590.24	8,128	0.811
1907–8.....	5,319.36	11,095	0.474
1908–9.....	6,451.62	11,084	0.582
1909–10.....	10,032.37	8,260	1.214
1910–11.....	5,842.50	9,022	0.648
1911–12.....	8,871.51	7,639	1.163
1912 to May 1	3,762.80	2,200	1.210
Total.....	$50,865.54	64,594	.....
Average per ton blister....	.....	.....	$0.787

In order to give some idea of the importance of the bag-house in connection with the converter establishment I present a number of pertinent data, and finally the financial outcome for the fiscal year 1910.

These data have been kindly furnished by Judd Stewart, General Auditor, American Smelting & Refining Co.

Leady Matte to Converters.

	Matte.	Iron.	Coppery Litharge.	Ore.	Flux and Linings	Total.	Slag.	Cleanings.	Fumes.	Total.
	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
Basic lined.....	11,885	167	2,442	668	1,890	17,034	7,765	1,964	1,253	28,016
Acid lined.....	9,659	.....	2,496	.....	2,671	14,826	6,427	3,443	1,055	25,751
	21,544	167	4,920	668	4,561	31,860	14,192	5,407	2,308	53,767

Contents.

	In Slag, Etc., Returned.		Total.	
	Lead.	Copper.	Lead.	Copper.	Lead.	Copper.
	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
Basic	4,369	5,766	3,233	595	7,602	6,361
Acid.....	4,309	4,176	3,003	649	7,312	4,825
	8,678	9,942	6,236	1,244	14,914	11,186

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Average Analysis of Matte.

Au.	Ag.	Pb.	Cu.	NiCo.	SiO_2.	Fe.	Zn.	S.	As.	Sb.
0.17	98.1	26.9	43.1	0.40	0.3	8.0	2.5	15.5	1.7	0.76

Fume Produced.				Saved in Bag-House.			
Part Returned to Converter				Tons 1,551–5,559.44	Ag at 0.54	$3,002.00	
	Tons.	Pb.	Cu.	2,650,774	Pb at 4.30	113,983.00	
				10,818	Cu at 0.08	865.00	$117,850.00
Basic....	1,849	1,115	.....	Direct Cost or Expenses.			
Acid.....	1,629	974	.....				
	3,478	2,089		Labor.	904.16		
				Power.	1,985.29		
				Sup. & Re.	2,953.05	5,842.50	
				Smelting 1,551 at $2.	3,102.00		
				Refining 1,325 at 5.	6,625.00	9,727.00	15,569.50
						Apparent net profit	$102,280.50

Less 6 per cent. interest on cost of Bag-house No. 1 (about $42,000.00).$2,520.00
2½ per cent. amortization..... 1,050.00 3,570.00

Net profit.....$98,711.50

No. 2. Blast-Furnace Bag-House System.

The blast-furnaces are served by two bag-houses similar to the converter bag-house, and while either can be cut out and cleaned, yet most of the time both houses are connected up. The flue leading to houses has cross-sections as shown in Fig. 7. Length of flue is 1,400 ft. Fan is 14 ft. in diameter with 8 ft. 11 in. inlet, running 140 rev. per min., and requires 62 h-p. The following figures give an idea of the duty of the blast-furnace bag-houses:

Period from May, 1909, to September, 1911, Inclusive.

Fume collected in No. 2 house, 1,275 tons..... 50 per cent. Pb.
Fume collected in No. 3 house, 1,348 tons..... 50 per cent. Pb.

This carried about 1,300 tons of lead. The lead-charge to the furnaces during this period was:

	Tons.
Lead in matte.....	7,287
Lead in ores.....	492
Lead in refinery by-products.....	23,475
Total.....	31,254

Of this 1,300 tons, or 4 per cent., went to the bag-houses.

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An analysis of raw fume before sintering gave: Pb, 52.5; Cu, tr.; Zn, 3; S, 5.4; As, 14.2; Sb, 1.6 per cent.; Te, tr.; Ag, 10 oz. per ton.

This raw fume contains a large amount of carbon and As_2O_3 and occasionally it ignites by spontaneous combustion and shrinks considerably in bulk.

In sintering, the arsenic not already present as As_2O_3 is oxidized and a portion volatilized. The fume sinters lightly and assumes a structure somewhat similar to coke. The arsenic is concentrated to about 20 per cent. by this operation and is found in beautiful crystals as As_2O_3.

Some experiments were made on roasting burnt fume to see how far the arsenic could be eliminated. The following results were obtained, roasting 24 hr. without fusing:

1. Original.....13.1 per cent. As.
   Final.....3.6 per cent. As.
2. 3,000 lb. of fume, 1,000 lb. of coal.
   Original.....13.1 per cent. As.
   Final.....3.7 per cent. As.

The quantity of burnt fume taken from house since 1906 was:

August, 1906, No. 2	1,200 tons.
June, 1908, No. 3	1,380 tons.
May, 1909, No. 3	200 tons.
May, 1909, No. 2	1,150 tons.
October, 1911, No. 2	1,270 tons.
December, 1911, No. 3	1,498 tons.

There is also some fume collected in the last 300 ft. of flue, to the extent of 280 tons per year.

The blast-furnace flue-dust taken from the flue (exclusive of the last 300 ft. mentioned) has been

1905–6	453 tons.
1906–7	666 tons.
1907–8	1,180 tons.
1908–9	944 tons.
1909–10	471 tons.
1910–11	714 tons.

The flue-dust is pugged with about 5 per cent. of lime and smelted in the blast-furnace.

The bulk of the bags in these houses generally lasts for several years and is replaced when fume is burned. This has been

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as long as three years, filtering out 1,498 tons of fume. Of course a few bags may have to be replaced in the interval.

The burnt fume carrying, say, from 18 to 20 per cent. of As and 2 per cent. of Sb is fused in a small reverberatory furnace and the gases led to a 300-ft. stack. This gives a slag, a mixture of oxides of Pb, Sb, and As, with about 10 oz. of Ag per ton. The arsenic remaining is about 5 per cent. A large amount of As_2O_3, very pure, could be recovered here, but the market does not warrant the necessary expenditure.

The fused material is allowed to cool in pots and is eventually charged alone into the residue-furnace of the lead-refinery (with a little coal if necessary), reducing a little lead from it to remove the silver. The antimonial slag thus produced (called "smoke" slag at the works) carries about 5 or 6 per cent. of Sb and 3 or 4 per cent. of As.

This slag, when reduced in a blast-furnace with lead-skimmings, makes a metal too high in As and Sb to make softening practical and is hardly a fit metal to dispose of as antimonial lead, as the As is about 3 per cent. and the Sb less than 10 per cent. Therefore the As is removed by a special process of fusion.

Blast-Furnace Bag-House Operating-Costs.

1905–6	$7,990.28
1906–7	13,180.49
1907–8	10,638.73
1908–9	12,903.21
1909–10	10,347.43
1910–11	6,557.54
1911	8,228.45
1912 to May 1	4,068.74
Total	$73,914.87

During the period 91,850 tons of lead were treated in the blast-furnaces, making the average bag-house cost per ton of lead treated $0.847.

During this same period the total lead treated in the entire plant was 810,082 tons; 0.0122 per cent. of this lead found its way to the bag-houses, the saving thereon being 42 cents per ton of total lead treated in the entire plant.

When the burnt fume from the bag-houses is fused, there is a material resembling speiss which separates out and freezes to the bottom of the furnace. It can be smelted out by increasing the fire. Some of it is tapped with the slag and forms buttons in the bottom of the pot. The composition is as fol-

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lows—evidently mainly lead telluride: Pb, 62; Cu, 3; Fe, tr.; S, 2; As, 4; Sb, 2; Te, 19 ; Se, 2 per cent.

Pure tellurium is recovered from this material by a chemical process. The market for this metal is very small and unsatisfactory.

Sack Record.

Converter Bag-House No. 1.		Blast-Furnace Bag-House No. 2.	
		1st Bag-House.	
July, 1905	960	Oct., 1906	1,040
Dec., 1905	940		
Feb., 1906	940	Oct., 1908	799
Sept., 1906	940		
Nov., 1906	960	Aug., 1909	1,020
May, 1907	860		
Sept., 1907	905	Dec., 1911	595
Feb., 1908	905		
May, 1908	978	*2d Bag-House.*	
July, 1908	1,040		
Dec., 1908	931	July, 1906	1,020
Mch., 1909	940		
June, 1909	940	June, 1908	1,062
July, 1909	940		
Aug., 1909	940	July, 1909	1,020
Feb., 1910	940		
Aug., 1910	940	Oct., 1911	690
July, 1911	1,880		
Dec., 1911	940		
Apl., 1912	940		

As some measure of the financial outcome of the operation of the bag-houses used in connection with the blast-furnaces (B.H. System No. 2) I give the following data kindly furnished me by Judd Stewart, General Auditor, American Smelting & Refining Co. These data are purposely given for a year in which less than half as much material was treated in the furnaces as is usually the case. This is done to show that even under very unfavorable conditions, the gain is satisfactory.

Material to Blast-Furnaces, Fiscal Year 1910–1911.

	Tons.
Copper by-products	23,090
Ore	22,762
Lead-refinery by-products	13,710
Matte	2,672
Lime-rock	6,345
Burnt lime	39
Iron	1,391
	70,009
Blast-furnace by-products	4,103
	74,112

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Fume Collected in Bag-Houses.

580 tons, containing 1,615 oz. Ag at 54c.		$872.00
460,000 lb. Pb at $4.30.		19,780.00
		$20,652.00
Expenses.	$6,558.00	
Smelting 580 tons at $2.00.	1,160.00	
Refining 230 tons at $5.00.	1,150.00	
		8,868.00
Apparent net profit.		$11,784.00
Less 6 per cent. interest on the cost of the two bag-houses, about $60,000.	$3,600	
2.5 per cent. amortization.	1,500	$4,100
Net profit.		7,684

(The works being a refinery centrally located, its life is assumed to be twice as long as that of the Murray plant, which has only a few mining-districts to draw from.)

No. 3. Zinc Oxide Process Bag-House.

The fume from the zinc oxide furnaces which treat certain by-products of the refinery is led through 100 ft. of brick flue and 500 ft. of steel flue, same as converter-flue, to a bag-house of similar constuction, having 120 bags, a 60-in. fan, using 15 h-p. for draft. Gases enter the bag-house at from about 190° to 200° F. This bag-house filters about 40 tons per month of crude zinc oxide, which contains about: Zn, 53.0; Pb, 20.0; Sb, 1.5; As, 0.3 per cent.

Several chemical methods have been proposed to convert this fume to pure ZnO, none of which have been put in operation as yet.

Fume from the cupels was at one time passed through a bag-house, but a bagless chamber was found sufficient to collect the values, therefore the use of the bags was discontinued.

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Development of the American Water-Jacket Lead Blast-Furnace.*

BY R. C. CANBY, WALLINGFORD, CONN.

(Cleveland Meeting, October, 1912.)

THE American water-jacket furnace is the outgrowth of lead-smelting at Eureka, Nev., subsequently developed in Utah and Colorado. Early smelting in Virginia, New England, or the Missouri-Kansas-Illinois district seems to have had no direct influence in the development of this furnace.

First Decade, 1870 *to* 1880.

In 1870, the draft-furnace was used in Nevada, coincidentally with the first crude sandstone blast furnaces, which were 2 or 3 ft. sq., with Sturtevant fans or small Roots blowers, and sheet-iron or water-cooled tuyeres. Stetefeldt, Liebenau, Albert Arents, and O. H. Hahn were the early metallurgical leaders. Arents's siphon-tap and the re-establishment of the furnace-bosh were the most important improvements of the decade.

Second Decade, 1880 *to* 1890.

At first there were many small local plants; but by the time Eureka had reached its maximum production of lead-ore, two custom works (the Eureka and the Richmond) were treating all the ore, the miners having learned the economy of smelting at larger and well-constructed plants. In the early eighties, most of the features of modern practice were already discernible at Salt Lake, Leadville, Denver, and Pueblo. The small square sandstone furnaces had been replaced by rectangular water-jacket furnaces, mostly 36 in. wide, and from 60 to 80 in. long at the tuyeres, which entered between cast-iron water-jackets. Subsequently, Franz Casin, at the Holden works, Denver, put the tuyeres in the middle of the jackets, and many other improvements were made by him and others, so that the cast-iron jacket became the favorite, until the possibility of

* Abstract of the paper, presented also (by title) at the Joint Meeting of the Institute with Section IIIa, Metallurgy and Mining, of the Eighth International Congress of Applied Chemistry, New York, September, 1912, and here published by mutual agreement.

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cheaper manufacture of steel-plate jackets made the use of the latter possible.

After the smelters had begun to do custom-work, paying cash for the valuable contents of the ores, as shown by assay, it became necessary to provide means for separating matte and speiss, which heretofore had been wasted with the slag. Frederick Overman[1] had set forth the importance of combining ores and fluxes to produce suitable slags; but the first systematic attempt to accomplish this was made at the Germania works, near Salt Lake, where Eilers and Hahn founded a new practice in lead-smelting. For a time great stress was laid upon exact formulas of composition, and definite "types" of slag. The larger furnaces, increased blast-pressure, and prepared ores of later periods have apparently permitted some relaxation in these rigorous rules.

At first, the slag was drawn into hemispherical pots, mounted on wheels, and the entire contents of the pot was thrown over the slag-dump. But in the early eighties attempts were made to separate matte and speiss from the molten slag. The conical overflow-pot, which, when partly filled with matte, was removed —another overflow-pot receiving the slag—came into use, as did also the continuous-flow fore-hearths, such as were adopted by Emrich, Mathewson, Iles, and Huber, and the Devereux pot, with a tap-hole in its side, allowing the retention of a slag shell for remelting. This led to the large double-bowl slag-car, drawn by mule on train-tracks. The Omaha & Grant works, Denver, had a system of large settling-pots, with Devereux tap-holes. Experiments in centrifugal separation at the same works, which failed mechanically, indicated that the silver in the slag was largely in the form of sulphide, which was mechanically disseminated. After centrifugal separation the slag was practically free from silver.

At this period, especially in Leadville, the mattes which were becoming so troublesome were iron-mattes, with little or no copper. Iron-matte in such small quantities in slags with increasing zinc-, barium-, and alumina-contents was difficult to separate from the slag, so that it was just at this period, the latter half of the second decade, during the transition-stage from carbonate to sulphide ores, that probably the greatest silver-losses were experienced.

[1] *Treatise on Metallurgy* (1852).

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It was a time also of great competition for supplies of carbonate lead-ores for fluxing, so that the obtainable percentage of lead upon the furnace-charge was forced to the lowest limit, much lower than was compatible with good metallurgical results. This aggravated the loss due to the disseminated-iron sulphide through the slag. It was still a period of high-iron slags, the specific gravity of which was by no means lessened by the increased percentage of barium coming from newly-developed mining-districts. This was probably the most trying period, both metallurgically and financially, through which the custom smelting companies passed.

Third Decade, 1890 *to* 1900.

By the beginning of this decade a metallurgical remedy had been found in the substitution for the small quantity of iron-matte of a larger quantity of copper-matte, which cleansed the slag. All the furnaces showed a "matte-fall" of from 5 to 10 per cent. of first matte, carrying from 2 to 8 or even (for short periods) 12 or 15 per cent. of copper. But where previously the slags had assayed from 0.2 to 0.5 and the mattes from 5 to 8 per cent. of lead, the slags now carried about 1.5 and the mattes from 15 to 20 per cent.

Furnaces had increased in size to about 42 by 120 in., and the blast-pressure was correspondingly increased. (Somewhat later, 1895–1905, there was a still greater tendency to force the blast to a maximum. At two works piston blowing-engines replaced the rotary blowers.) High blast developed channeling, slips, etc., and drove the smelting-zone up so high in the shaft that the furnaces burned out over their water-jackets and water-cooled linings were resorted to, while the lead reduced in the upper zones was reoxidized at the tuyeres and went into the slag and matte.

Studies of the form, bosh-inclination, etc., and of the composition of escaping gases, yielded some interesting information, but no practical remedy for the improper reduction. It become evident that unsatisfactory furnace-conditions were very largely due to the fineness of the material charged, and attention was directed to the advantages of "pot-roasted" or otherwise sintered ore. The very general use of such predi-

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gested furnace-feed has been probably the most beneficial step of progress in lead-smelting.[2]

It is a noticeable fact, if one glances over the American technical literature, which naturally reflects the point of view of the two branches of metallurgy, that one finds in the lead-smelting such a preponderance of consideration given to the slag-composition, or slag types, whereas regarding the iron blast-furnace cinder, scarcely a reference is made to its analysis. This, of course, is perfectly natural, since the conditions are so entirely different. The lead-furnace produces a great preponderance of slag, while in the iron-furnace the product, pig-iron, is relatively so much greater, and the iron has so high a specific heat and temperature that it renders quite negligible the question of cinder-composition. It is, therefore, the consideration of the gases, the relative proportions of carbon monoxide and carbon dioxide, which has more largely occupied the attention of the iron-furnace metallurgist.

The preliminary roasting of ores in the hand reverberatory furnaces had about reached its greatest perfection with the beginning of our third decade (1890), and the "fusion-box," as an adjunct to the hand-roaster, had practically run its somewhat costly course. Early in this decade came an era of large mechanically-operated roasters of a reverberatory type.

The Fourth Decade, 1900 *to* 1910.

This decade is characterized by larger furnaces and corresponding mechanical devices for handling material. The general practice has been to gain hearth-area by adding length, the width of 36 in. being but slightly increased. In order that 3 or 4 instead of from 7 to 12 furnaces should treat all the ores at a given plant, every adjunct of the furnace had to be correspondingly enlarged. The slag-cars, carrying from 3,000 to 8,000 lb., were operated in many cases by steam or electric motors. (I used at the Arkansas Valley works, Leadville, in 1888, the first locomotives for hauling lead blast-furnace slag.) For the more perfect separation of the matte, large reverberatory forehearths, holding from 30 to 40 tons (the Rhodes-Van Cleve separator) were introduced; and at El Paso I added

[2] For particulars concerning blast-roasting, see the article of A. S. Dwight, Efficiency of Ore-Roasting, *Engineering and Mining Journal*, vol. xcii., No. 27, pp. 1267 to 1271 (Dec. 30, 1911).

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an arrangement for granulating the matte direct from the fore-hearth. But the reverberatory slag-separator required a suitable slag, and hence limited the amount of zinc permitted in the charge.

Matte-settlers or fore-hearths constructed with sectional sides and ends (which can readily be taken apart for removing crusts) and mounted on trucks to fit the matte-car tracks are now generally used.

The mechanical feeding of the furnace was another step in advance. The bell-and-hopper had been used on round furnaces, but tended to give too hot a furnace-top. The first successful mechanical charging was done at the plant of the Pueblo Smelting & Refining Co. in 1895.

Later, Dwight modified Hixon's East Helena automatic feeder by adding the baffle-plate device for throwing coarser material to the middle, and finer towards the side-wall, which has since become the essential feature of automatic feeding.[3] He was the first to emphasize adequately the importance of the mechanical condition and distribution of the charge, and to point out the great benefit of a proper preparation of the material.

The modern lead-furnace, generally speaking, is from 45 to 48 in. wide and from 160 to 180 in. long, with a daily capacity of about 200 tons. At one plant, 48- by 160-in. furnaces had a capacity of from 140 to 160 tons, which was increased by using a proportion of Huntington and Heberlein pot-roasted product to about 200 tons, and afterwards, by the Dwight and Lloyd sintering process, to 250 tons or more. The same furnaces running on matte-concentration have a capacity of from 350 to 375 tons. The largest lead-furnace in operation is 42 by 220 in. in size.

Comparisons of the cost of smelting and metal-losses at the different periods are impracticable or would be misleading, by reason of the varied conditions, explained in the full text of this paper.

The subject of the collection of flue-dust and condensation of fumes will not be considered in this paper, in view of the paper of Anton Eilers, Notes on Bag-Filtration Plants, presented at this meeting.

[3] See The Mechanical Feeding of Silver-Lead Blast-Furnaces, by A. S. Dwight, *Trans.*, xxxii., 353 to 395 (1901).

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The Development of the Parkes Process in the United States.*

BY ERNST F. EURICH, NEW YORK, N. Y.

(Cleveland Meeting, October, 1912.)

ALEXANDER PARKES patented in England in 1851–52–53 a process for desilvering lead by means of zinc, making use of the greater affinity of silver for zinc than for lead, discovered by Karsten in 1842. The efforts in England to develop it commercially were not particularly successful, nor was the attempted introduction into the United States. This last was not surprising, as it was about the beginning of our Civil War, and at that period there was very little lead-bullion produced in the United States and there was no field for the process here. There was at that time on Staten Island an establishment that pattisonized lead, probably from Rossie, N. Y., and Chester, Pa., and possibly some Spanish lead, but the production of lead-bullion in the Far West had not yet begun. In 1864, Edward Balbach, of Newark, N. J., patented in this country a process for desilvering lead by means of zinc, and in 1867 Edward Balbach, Jr., patented a "movable black-lead retort with a neck, placed in a furnace," for the purpose of distilling the alloy of zinc, lead, and the precious metals.

A brief comparison of the Parkes and the Balbach processes is interesting. The Parkes process as practiced in England consisted in stirring into silver-bearing lead, melted in a kettle, from 1 to 2 per cent. of zinc, cooling, skimming the alloy of lead, zinc, and silver that rose to the surface, liquating it in an iron retort, distilling the residual rich alloy in fire-clay retorts, thus regaining the zinc, and refining the desilvered lead at a low heat in a shallow cast-iron pan that formed the hearth of a reverberatory furnace. If the lead was too impure, it was first softened

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in a reverberatory furnace similar to the refining-furnace. The Balbach process liquated the softened bullion in a reverberatory with a cast-iron inclined hearth, the eliquated metal flowing into a kettle which contained the requisite quantity of melted zinc, thoroughly stirred the contents and ladled into molds, transferred the bars to a second liquating-furnace, where they were heated to the melting-point of lead. The eliquated lead was to be measurably free from silver, the residual rich alloy was distilled in a black-lead retort, the zinc given off condensed and the rich lead resulting cupelled. The desilvered lead was refined at a high heat in a reverberatory built of fire-brick. Impure lead-bullion was first softened in a reverberatory furnace similar to the one used for refining. The Balbach process as patented had some of the features of the Parkes, and after several years of operation, about 1872, was so modified as to resemble it more closely.

It is interesting to note that in the desilvering of lead-bullion by means of zinc as practiced to-day, the different operations are fundamentally identical with those outlined by Parkes. The failure of the process, or rather the lack of commercial success, was due solely to the apparatus employed; the shallow cast-iron pans frequently cracked, and the fire-clay retorts were not adapted to the work required of them. By the use of fire-brick furnaces for softening and refining, and of black-lead retorts for distilling, those works which later, in the United States, followed the lead of Parkes in their operations, made the Parkes process an unqualified success. This is the process by which most of the lead-bullion in the civilized world is desilvered, and in many respects it has found its best developments in American plants.

With the opening of the Pacific railroads, the exploitation of the Far West received a great impetus. Furnaces for smelting lead- and silver-ores were soon in operation, and the lead-bullion they produced, as well as some of the ore mined, sought a market in the East. Smelting- and refining-works were built at different points from the Atlantic to the Pacific, and by 1873 there were 14 establishments in operation at which smelting of ores and refining of lead-bullion was carried on. Two of these, the Pennsylvania Lead Co., near Pittsburg, Pa., and the St. Louis Smelting & Refining Co., near St. Louis, Mo., certainly

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the former, followed exactly the lines laid down by Parkes, using, however, black-lead retorts and fire-brick softening- and refining-furnaces. The Germania works at Salt Lake City soon followed suit, and of the remaining refineries, the Balbach works at Newark, N. J., the Omaha works at Omaha, Neb., and the Selby works at San Francisco were soon the only survivors. These three used the Balbach process, but it had been so modified as to approach closely to the Parkes model. By the end of the 70's or the early 80's various processes had been tried out, and only the Parkes remained.

However much the American works differed in the details of operations, there was one feature that soon became common to all, the terrace construction of the desilvering-plant. The softening-furnace occupied the highest level, the desilvering-kettle the next, the refining-furnace below this, and lowest of all was the floor on which were the molds for the merchant lead. The charge finished in one apparatus was tapped and flowed into the next, and from the last was run into the molds.

To trace the improvements in the construction and operation of the constituents of the plant, it will be simplest to consider each operation separately.

Softening Lead-Bullion.—This operation is carried out in a reverberatory furnace. We early find the hearth of the softening-furnace built inside a riveted sheet-iron pan, the material of the hearth being fire-brick. In this construction, in use in 1873 and possibly earlier, the wrought-iron prevents all leakage of lead out of the furnace, and the bottom, of fire-brick, laid as an inverted arch of from 6 to 9 in. rise, effectually resists the plumbostatic pressure of the charge. Adopted later in Europe and wherever the Parkes process is used, it has become the standard construction.

To counteract the corroding action of the oxides at the lead-line, and reduce the frequeney of repairs, water-cooling in various forms was resorted to. A 2.5- or 3-in. water-pipe was placed along the lead-line next to the pan, or the sides and ends of the hearth were provided with water-jackets, or again the pan proper was placed in a second riveted pan, leaving a space between the sides and bottom in which water circulated. This last was eventually abandoned, as the cooling of the bottom was not required and increased the fuel-consumption. With the

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advent of magnesite bricks, these were used along the lead-line in addition to the water-cooling. Indeed, in some cases they have led to the abandoning of water-cooling except under the skimming-door.

With the increase of the regular supply of lead-bullion that could be depended on, the size of the softening-furnace increased from a capacity of 10 tons to 25, 40 and up to 90 tons. With hand-charging this was about the economical limit, but the introduction of a charging-apparatus (first used at the National works of the American Smelting & Refining Co.), which by means of an endless belt elevated the pigs and dropped them into a chute down which they slid into the furnace through a hole in the roof, has made still larger furnaces feasible, and a from 210- to 240-ton furnace is now in operation at Omaha. In the matter of size of furnaces, England was for quite a time in the lead, as 140-ton furnaces were in use there for a number of years before furnaces of a similar size were built here. The time required for refining a large charge in a properly proportioned furnace is no longer than is required for a smaller one, and, provided sufficient lead-bullion is available to keep it in steady operation, a large furnace is more economical in labor, fuel, and repairs than a smaller one.

The proper method of conducting softening-operations has long been known, but not always properly observed. When its laws were transgressed the penalty followed,—a smaller direct yield of soft lead, a larger quantity of secondary products to be reworked, and greater costs. It is not necessary to go minutely into the details of the operations. The important points are: melting the charge at a low heat and removing the dross that accumulates on the surface, which contains the greater part of the copper present; next, increasing the heat, admitting air, oxidizing the impurities susceptible of oxidation (mainly arsenic and antimony) and skimming off the arsenical and antimonial lead oxides. In some works crushed litharge free from copper was regularly used for the purpose of oxidizing the impurities. The advantages of reducing the lead of the litharge and at the same time oxidizing the impurities are evident. It is quite noticeable that as soon as the arsenic is pretty well eliminated, the oxidation of the charge proceeds more rapidly, and

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it is for oxidizing the arsenic that litharge has been found particularly helpful.

For a time the lead-smelter in the West, with few exceptions, loaded into the lead-bullion which he shipped, as much as possible of the dross the furnace made. Gradually it was appreciated that this procedure was not to his advantage, and it was modified. The Philadelphia Smelting & Refining Co., at Pueblo, Colo., was the first systematically to prepare clean bullion for shipment. It was drossed in the cooling-pot before ladling, the resulting pigs were remelted in a large kettle holding a carload and, after careful drossing, run into molds. At the furnaces controlled by the American Smelting & Refining Co. the same practice is now followed. Such preliminary drossing, carefully carried out, renders drossing in the softening-furnace unnecessary, and softening then consists merely in eliminating the arsenic and antimony. After softening, the charge is tapped into a launder, through which it flows into the desilvering-kettle.

Desilvering With Zinc.—The kettles employed for this operation originally had a capacity of 10 tons, but this capacity has been increased to 60 or 70 tons. At first kettles were cast with a spout at the bottom through which the charge was tapped into the refining-furnace. In 1877, at the works of the St. Louis Smelting & Refining Co., the use of a siphon for transferring the charge from the desilvering-kettle was introduced by A. Steitz. His example was followed by other works and the spout, no longer necessary, disappeared. In recent years a horizontal centrifugal pump driven by an electric motor has come into use for emptying the kettles. With large kettles, the centrifugal pump saves time. Furthermore, its use will affect the design of the kettles and will have an influence in the arrangement of the constituent parts of plants that may be erected in the future, and may use it. No longer limited to a depth of 40 in., the kettle can be made deeper and capacities in excess of 60 tons will be entirely feasible.

The apparatus for liquating silver crusts varied at different works. At some it was a reverberatory furnace with an inclined cast-iron hearth; at others, a kettle with or without a spout. In the former case an additional kettle was provided to receive the eliquated lead. These various devices have been

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superseded by the silver-crust press, which, producing a richer crust, is now in general use. It was first introduced at the works of the Pueblo Smelting & Refining Co. by W. H. Howard, its inventor and patentee.

It is of prime importance that the zinc which is to effect desilvering be well incorporated with the lead-bullion. At first this was done solely by manual labor. In the later 70's steam-stirring was introduced at the works of the Pennsylvania Lead Co. It consisted in passing dry steam through the charge, producing a circulation which, combined with a rabbling of the surface, effected a good incorporation of the zinc which had been previously melted on the surface. Both of these methods have given way to the mechanical stirrer, invented by W. H. Howard and patented in 1894 and first used by the Pueblo Smelting & Refining Co. Like its prototype, the device of Cordurié, it uses a propeller-wheel for establishing a circulation and incorporating the zinc. The details of its construction, however, make it more efficient. It works by steam or electrical power and effects a thorough and uniform mixing of the metals.

For quite a time there was no material change in the manner of conducting zincings from that practiced in the early trials of the Parkes process in England. The general custom was to add the zinc in three doses, stirring in each addition, and then cooling and skimming until a ring of chilled lead had formed around the kettle. Only the first crust was liquated and the dry crust sent to the retorts; the second and third crusts were used again in the next charge. Some works used four zincings, the first to collect the gold along with a small part of the silver in a so-called gold crust. When this was done, the first and second crusts were liquated separately and the dry crusts were worked up each for itself.

With a wide range in the silver-content of the lead-bullion received from various sources, it was desirable to have tables for the zinc additions. Lead desilvered by zinc contains at its melting-point 0.6 per cent. of zinc. The total quantity of zinc required to extract the silver from any given quantity of lead-bullion is represented by a formula containing a constant (0.6 per cent. of the weight of the charge) plus a variable that increases with the silver-content, but at a lower ratio. On this

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basis, tables for zinc additions were worked out covering a wide range of silver-content.

The time required for desilvering a charge was from 15 to 20 hr. Attempts to reduce the time by reducing the number of zincings were successful, but at the expense of larger zinc additions. That such was the case was due to the incomplete incorporation of the zinc. With the use of the Howard stirrer, supplemented by the Howard press, the total time of desilvering has been reduced to not to exceed 10 hr. The zinc additions are not greater and there are no longer coolings to a "ring," until the very last, when the silver-content is to be brought down to the assay required in the merchant lead.

Refining.—Eliminating the zinc retained by the desilvered lead has, in the United States, been effected either in a reverberatory by oxidation at a comparatively high heat, or in a kettle, by passing dry steam through the lead previously heated to redness, as was first practiced by Cordurié. Steaming in kettles was used by some of the early unsuccessful works; it has been repeatedly tried experimentally, the Pueblo Smelting & Refining Co. alone adopting it permanently for refining. The use of the reverberatory has found most favor with American refiners, and at the present time it is exclusively employed.

The construction of the refining-furnace is exactly the same as that of the one used in softening. Its size has kept pace with that of the softener, so that while the charge of a large softener is divided among two or more desilvering-kettles, the contents of these kettles are again gathered together for treatment in one large refiner.

After the zincky oxides have been skimmed, the ordinary practice is to raise the heat for a short time and to admit air, in order to remove the last of the antimony. At some works this final removal of the antimony is accomplished by steaming for a short time without raising the heat. The same procedure has been employed in accelerating the elimination of the antimony in the softening-operations.

Tapping the refined lead into a merchant kettle and ladling into molds were early replaced by more expeditious methods. In 1876, at the Pennsylvania Lead Co.'s works, this was done by running the lead through a swinging 1.5-in. pipe into molds arranged in the segment of a circle, the flow being controlled

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by a cast-iron cock. With the introduction of the siphon, some works continued to use the merchant kettle and employed the siphon to run the lead into molds.

The next improvement was the molding-machine, a moving endless belt on which the molds are mounted. As each mold comes under the discharge-pipe it is filled, is then carried forward, cooled, and as it passes over the end sprocket-wheel the pig of lead is dumped. This machine, introduced at the National works by the then superintendent, A. Oechsler, is especially adapted for handling large charges economically.

Distilling the Zinc Crust.—The early use of the black-lead retort by Balbach has already been noted. It was mounted in an inclined position in a coke crucible-furnace, the neck of the retort projecting through the front wall, the zinc distilled off being condensed. When the charge was finished, the supporting front wall was removed, the crucible seized by tongs, turned down and emptied. A. Faber du Faur, in 1870, swung the crucible-furnace on trunnions, so that by tilting it the retort could be emptied. This tilting furnace is used to-day by all the lead-refiners in the United States, and has been largely adopted in Europe.

About 1865 the Parkes process was taken up on the continent of Europe. It is interesting to note the various ways that were followed in treating the zinc crust. At some works it was smelted in a blast-furnace and the resulting lead cupelled; at others it was scorified on a lead bath. It was also treated with steam to oxidize the zinc, the resulting oxides being scorified on a test or treated with acids. In all cases the zinc was sacrificed and there was an appreciable silver-loss. These various methods are in striking contrast to the simplicity of the original Parkes method made practicable by the American modifications.

From a capacity of 250 lb. the retorts were gradually increased in size until a capacity of from 1,200 to 1,400 lb. had been reached. As the tilting furnace was a patented device, it was not used by all refiners during the life of the patent. Various furnace constructions were devised, some using coke, others coal, and all using the same kind of retort, but instead of emptying the retort by tilting, the contents were tapped or dipped out. With the expiration of the patent, these various

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constructions were replaced by the Faber du Faur tilting furnace. The only change in the operation has been the substitution of fuel-oil for coke. The advantage is not so much in reduced fuel-costs, but rather in the longer life of the retort, the more uniform and better results obtained, and in the fact that the work is easier for the men.

Cupelling Rich Lead.—Although cupelling rich lead is not peculiar to the Parkes process, and, together with the treatment of the various secondary products, belongs rather to the general metallurgy of lead, yet as the large tonnage handled by American works using the Parkes process has brought about a characteristic development of this operation, a brief sketch of its development in this country will not be out of place.

In the United States, cupellation has always been effected in the English type of cupel-furnaces. Small at first, they have increased gradually from a test-capacity of 2,000 lb. or less to one of 6 tons. The use of bone-ash for making the test soon gave way to other, cheaper and more satisfactory, materials. Varying with individual preference, there have been used Portland cement, mixtures of Portland cement with ground fire-brick, Portland cement with ground limestone, ground limestone with fire-clay; and test frames have been lined with magnesite brick. At present, a mixture of crushed or ground limestone (preferably dolomite) with fire-clay is generally used. To counteract the corrosion of the test by litharge, and the eating out of the litharge channels, various water-jacket devices have been introduced; A. Steitz, at the St. Louis works, being the first to use water-cooling for this purpose.

The test of the English cupelling-furnace, while it is removable, is stationary when put in position in the furnace. American practice early developed a tilting test, and both stationary and tilting tests are at present in use. The latter is mounted in such a manner that, by means of a differential block or by a hand-wheel, it can be tilted up or down and the flow of the litharge controlled. The prototype of this device is found in a silver-refining furnace in use in Sweden at Konigsberg in 1854. The tilting test was especially convenient before the introduction of the litharge water-jackets, and even under present conditions is preferable to the stationary test.

For a long time it was customary to cast the rich lead from

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the retorts into bars and gradually feed these bars into the cupelling-furnace. A marked improvement was effected about 1901, when F. B. F. Rhodes, at the National works at South Chicago, poured the charge from the retort into a pot, raised the pot by means of a pneumatic lift and poured the contents into the test. This procedure has been followed with advantage at other works.

Thus the simple small English cupelling-furnace has been developed into its American descendant, which, in its most complete form, has a tilting, water-cooled test holding 6 tons or more of rich lead, is capable of oxidizing 7 tons of lead in 24 hr., and produces doré silver 998 fine. Depending on the method of parting to be used, whether by electrolysis or sulphuric acid, the contents are either ladled into thin slabs, or they are poured into molds by tilting the test.

Except in the case of the retorts, no reference has been made to the use of oil as fuel. In a general way it will suffice to say that where the price warrants, as at the Selby works at San Francisco, Cal., it has satisfactorily taken the place of coal.

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The Constitution and Melting-Points of a Series of Copper-Slags.

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(Cleveland Meeting, October, 1912.)

I. INTRODUCTION.

THERE are comparatively few accurate data on the melting- or the freezing-point temperature of metallurgical slags, or on related physical phenomena, such as fluidity near the melting-point, specific heat, latent heat, etc. With the exception of the work of H. O. Hofman,[1] and some older work by R. Åkerman, P. Gredt, and H. M. Howe, no information is available that is of direct interest to the metallurgist.

Hofman's valuable work is the only one that refers to slags produced in the metallurgy of copper and lead. The classic work of J. H. L. Vogt,[2] who views slags from the standpoint of physical mixtures (solutions in which the entities are minerals), has opened up a new field in the study of slags, and has placed the investigation of such properties as melting-temperatures, etc., on a rational basis. The recent work of A. L. Day, E. T. Allen, and W. P. White, of the Carnegie Institution, at Washington, and of European investigators[3] on certain systems of silicate minerals confirms the value of the investigation of silicate series from this standpoint, and it is my aim to apply this method to metallurgical slags, in order to gain exact data of value in metallurgy. The work recorded in the present paper is the result of an investigation of reverberatory-furnace slags made for a Western smelting company to determine exactly the influence of change in composition, within comparatively narrow limits, on the melting-point.

For a general discussion concerning the mineralogical character of slags and the principles on which this investigation is based, reference should be made to other work,[4] for the subject is too large to be stated in this paper, except very briefly in the following outline:

[1] *Trans.*, xxix., 682 (1899).

[2] *Die Silikatschmelzlösungen; I. Ueber die Mineralbildung in Silikatschmelzlösungen; II. Ueber die Schmelzpunkt-Erniedrigung der Silikatschmelzlösungen* (Christiania, 1903–04).

[3] Doelter, *Handbuch der Mineralchemie* (1911)

[4] C. H. Fulton, *Principles of Metallurgy.* Chapter viii., on Slags, p. 245 (1910).

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Copper-slags, broadly considered, may have this range in composition: silica, from about 30 to 50; lime (including some magnesia), from about 2 to 35; ferrous oxide, from about 12 to 65; alumina, up to 12 per cent. Other bases, such as magnesia and zinc oxide, may be present in considerable amount. If so, the subject requires separate discussion and it is again referred to. When present in small amount, both magnesia and zinc oxide may be considered to replace iron oxide in some of the minerals formed. With the above range in composition the silicate degree of the slag varies between an ortho-silicate and a meta-silicate, and for practically all slags lies somewhere between the two. The slags when molten will consist largely of solutions of two minerals, one an ortho-silicate and the other a meta-silicate. What these minerals are depends upon the base composition of the slag, very largely upon the relation of lime to iron oxide to alumina. In this manner series are formed in which the two minerals are respectively the entities or end-members, and all compositions between are mixtures, in various proportion, of the two minerals. The addition of one mineral to the other will cause a change in the freezing-point according to well-known laws, and for the range in composition from one mineral to the other there is established a freezing-point or a melting-point curve which is rational and will fall into one of the well-known types.

In Table I. are given the mineral series possible for copper- and lead-slags, according to silicate degree and base composition.

TABLE I.—*Mineral Series of Copper- and Lead-Slags.*

Silicate Degree.	Mineral Series.	Base Composition Within Limits Specified.
Ortho-silicate	1. Olivine—magnetite (fayalite)	Very high in iron.
Ortho-silicate	2. Olivine—melilite	High lime, comparatively low iron. High alumina.
Between ortho- and meta-silicate.	3. Melilite—augite	Same as above.
	4. Augite—olivine (iron-lime olivine)	Somewhat less lime, more iron; high alumina.
	5. Augite—olivine (fayalite)	Comparatively high iron, still less lime, lower alumina.
	6. Rhombic pyroxene-olivine (hypersthene)—fayalite	Very little lime, high iron low alumina.
Meta-silicate	7. Rhombic pyroxene (hypersthene)—augite	Low lime, comparatively high iron; some alumina.

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The slags ordinarily made in copper-smelting will fall chiefly into series 3, 4, 5, and 6, dependent on the base composition. Alumina in the presence of considerable lime will go to form augite or melilite. If the composition of the slag mixture is such that neither of these minerals can form, alumina is very apt to form spinel, which crystallizes out of the molten slag at a high temperature, causing difficulties. The formation of spinel is greatly aided by the presence of magnesia or zinc, or both, even in small quantities. Ordinarily magnesia passes to the olivine or hypersthene mineral, zinc to the olivine mineral, particularly to the high-iron olivine or fayalite. It is thus apparent that if a slag is to carry considerable alumina it must also contain considerable lime in order to form augite or melilite, and if it is to carry considerable zinc, much iron and but little lime must be present to give opportunity to form fayalite. High alumina and zinc are incompatible. High alumina and magnesia are incompatible.

It is to be noted that the minerals named are used to designate group minerals. Thus olivine may vary widely in composition, including such minerals as fayalite, true olivine, an iron-lime olivine, etc. The same is true of the other minerals named. For the purpose of a determination of melting-points and related properties it is proposed to investigate the series outlined in Table II.

TABLE II.—*Base Composition of Slags.*

Silicate Degree.	Base Composition.	Mineral Series.
Silica varying between ortho- and meta-silicate....	1. $\frac{CaO}{FeO} = \frac{70}{30}$ 2. $\frac{CaO}{FeO} = \frac{}{40}$ Alumina up to 18 per cent.	Melilite—augite.
Silica varying between ortho- and meta-silicate.....	3. $\frac{CaO}{FeO} = \frac{50}{50}$ 4. $\frac{CaO}{FeO} = \frac{40}{60}$ Alumina up to 8 per cen	Olivine—augite.
Silica varying between ortho- and meta-silicate....	5. $\frac{CaO}{FeO} = \frac{30}{70}$ 6. $\frac{CaO}{FeO} = \frac{5}{95}$ Alumina below 6 per cent.	Olivine —augite. —hypersthene.

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If the melting-points of a considerable number of slags in these six series be determined, the field of copper-slags will be practically covered, for additional curves of interest can then be plotted from the data obtained by taking the slags of the same silica-content from the six series, showing the effect of varying ferrous oxide and lime with constant silica. The effect of varying alumina will have to be separately determined.

In the work described in the succeeding sections of the present paper one olivine-augite series and two melilite-augite series were investigated, in which in each series the iron-alumina ratio remained constant, while the lime varied with the silicate degree. The reason for this is that the ordinary reverberatory-slags fall into such a series, and that the problem presented was a practical one, viz.: with a certain ore-mixture essentially constant in composition with regard to silica, iron, and alumina, what is the effect on the melting-point of the slag formed of the addition of increasing quantities of lime (the flux)? Also, how low may the lime be, and how high can the silicate degree be carried, and still have practical slags?

II. Constitution of Certain Reverberatory-Slags.

In investigating the problem it was thought desirable to examine the slags from the reverberatory furnaces from the chemical and mineralogical standpoints. Table III. gives in a concise way the compositions of these slags, which were produced by smelting calcines obtained by roasting concentrates mixed with crushed limestone, with a certain amount of converter-slag, from both basic- and acid-lined converters. The analyses represent very careful work with accurate and exact methods, not the methods used in ordinary rapid smelter-work. The regular slags tapped from the furnace are designated as "clean slags." "Floaters" are masses of loosened furnace-bottoms. "Blanket slag" is a slag which forms on the surface of the charge and which seems to consist of partly-fused material. The following additional petrographic notes elaborate the identification of the minerals. (I am indebted to Dr. Charles P. Berkey for the identification of the minerals in the slags).

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TABLE III.—*Composition of Reverberatory-Furnace Slags.*

No.	Type.	Composition.									Petrographic Notes.
		SiO_2.	Fe_2O_3.	Fe.	Al_2O_3.	CaO.	MgO.	Cu.	Zn.	K_2O Na_2O.	Total.[a]	
		P. C.	Per Cent.	P. C.	P. C.	P. C.	P. C.	P.C.	P.C.	P. C.	P. C.	
2	Blanket slag.....	38.68	42.36 $FeO = 38.10$	29.65	8.48	9.92	0.79	.....	.....	1.00?	96.97	Ground-mass essentially a brownish glass. Small feather-like crystals of a pyroxene (augite). Small pieces of unaltered quartz. Numerous matte globules. Flow structure.
3	Blanket slag.....	52.08	31.08 $FeO = 27.96$	21.76	9.00	5.90	0.23	.....	.....	1.00?	96.17	Main mass essentially a brown glass. Flow structure. Considerable magnetite. Some small feather-like crystals of augite. Unaltered quartz. Numerous matte globules.
9	Floater slag.....	84.62	9.48 $FeO = 8.53$	6.64	2.13	0.90	Tr.	.....	.....	1.00?	97.18	Ground-mass a glass. Much original quartz. Many rod-like colorless crystals, perhaps olivine? Some matte globules.
12	Clean slag.....	38.50	42.20 $Fe_3O_4 = 40.80$ $FeO = 37.98$	29.54	7.62	12.62	1.01	0.21	.....	1.74 0.50	99.97	Ground-mass a glass. Large number of feather-like crystals of a pyroxene, probably hedenbergite. Some colorless rod-like crystals, as above. Considerable magnetite. Matte globules.
13	Clean slag.....	42.74	35.94 $FeO = 32.33$	25.16	7.76	13.32	0.94	.....	0.60	1.00?	98.09	Mostly an aggregate of green fern-like crystals of a pyroxene (augite), constituting about 75 per cent. of the slide. Considerable magnetite. Numerous very small matte globules.
14	Clean Slag.....	40.12	40.47 $FeO = 36.40$	28.33	7.21	13.22	0.98	.....	.....	1.00?	98.93	Same description as for No. 13.
15	Clean slag	37.24	47.68 $FeO = 42.92$	33.38	6.38	10.32	0.96	.....	.....	1.0 ?	98.82	Mostly an aggregate of light green radiating, interpenetrating crystals of probably hedenbergite and a darker light brown augite. Much magnetite, some matte globules.
16	Crystalline slag...	49.96	29.67 $FeO = 26.69$	20.77	8.90	12.12	1.30	.....	.....	1.00?	99.97	Consists largely of augite, well crystallized, the balance being magnetite and a little glass.

[a] Exclusive of Zn, Cu, and S. Iron as FeO.

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Slag No. 3.—A dark slag with many roundish light-colored grains. One mineral is present in considerable amount. It is colorless and occurs in round, corroded grains that have neither crystal-boundaries nor internal structural features. The grains are fractured. The mineral has good relief and low double refraction. It is uniaxial and positive. It is evidently the original undecomposed quartz that has not been fluxed. The ground-mass is glassy, shows strong flowage, and is brown to black in color. Those portions showing the strongest flowage are brown in color, and portions immediately surrounding the crystals grains are generally brown. The rest of the field is very dark and carries an abundance of black metallic particles, very likely magnetite. The lighter color about the quartz grains seems to show the solution of the quartz into the glass or slag. In a few places large metallic grains occur, probably matte globules. Certain areas show no flowage at all, but have a very minute micro-granular structure that is the beginning of crystallization. In limited areas this gives a mottled effect similar to that produced by the devitrification of natural glasses. These areas have a yellowish or greenish-yellow color, and a refraction and double refraction of higher order than other areas. They are probably the beginnings of pyroxenes.

Slag No. 9.—An abundance of crystalline minerals is present. The minerals grow in rods, or plates, which are in bundles or aggregates, and constitute about one-third of the area. It is of moderate relief, entirely colorless, strong crystallizing tendency, of low double refraction, parallel extinction in all cases, uniaxial and positive. The latter characteristics are not very readily made out, as the interference figure is indistinct. The elongation is parallel to the "a" elasticity axis. The slag has a striking structural appearance. The rod-like structure is strongly developed, with numbers lying parallel and with inclusions of the dark glassy matrix in and between them. They are associated with other groups lying at various angles and larger aggregates of less definite boundaries. The mineral is not identified. The rest of the slide consists of comparatively large areas of original quartz, unfluxed, and a greenish-brown glass. Minute particles like mineral dust are also present.

Slag No. 12.—The main body of this slag is divided between an opaque glass and a glassy green mineral. In very thin sec-

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tion the color is green to brown, and the mineral is pleochroic. The extinction angle when measured from the central shaft of a fern-like crystallite is often high, up to 30°. Refraction and double refraction are strong. Interference colors are of the second and third orders in thin sections. There is little doubt that the mineral is a pyroxene, probably near hedenbergite. In some of the better formed crystals there are lines of opaque inclusions of a yellow-gray and a steel-gray metallic substance, probably magnetite and sulphides (matte). Magnetite also occurs in little patches with rounded irregular outline, forming a considerable proportion of the slag (perhaps from 5 to 10 per cent.). There are some oblong crystals of a nearly colorless mineral with cleavage parallel to the elongation and extinction parallel to the cleavage. This corresponds to a member of the olivine group, although the double refraction seems to be less than in natural olivine. The slide tested with acid shows gelatinization, pointing to the presence of olivine. The rod-like crystals are of the same kind as in slag No. 9.

Slag No. 14.—In this slag there is enough crystalline structure to show that some mineral constituent is very generally developed in the slag. The structure shows itself in branching and feather-like aggregates, and in areas where the components have strong individual elongation. The mineral is light green in color. It shows brownish pleochroism and an abundance of dark inclusions which accentuate the structure. It is biaxial with oblique extinction and high double refraction and strong relief. The extinction angle varies from zero to more than 40°. These are the characteristics of the colored monoclinic pyroxenes. The color undoubtedly indicates that iron is a prominent constituent. The mineral corresponds more nearly to common augite than to any other variety. No other mineral is developed in this slide except magnetite, which is plentifully distributed about the branching forms of the augite. This last constitutes about 75 per cent. of the slag.

Slag No. 15.—This slag shows crystalline development in long needles of radiating, branching and interpenetrating structure, and the matrix between them shows also the characteristics of the incipient crystallization. The crystals and aggregates are light brown to light green in color. The only other individualized constituent occurs as minute, metallic

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black grains, some of which are apparently octahedral in form and very abundant throughout the section. This mineral is magnetite. The light-colored crystals appear to be of two distinct varieties. The one most strikingly developed is a very light green, and is in the form of long needles with feather-like branches or cross-bars. It is always surrounded and intergrown with an excessive amount of black granular material, probably magnetite, which is more abundant in its immediate vicinity than in other parts of the section. This mineral is a pyroxene, and so far as tests can be made agrees well with hedenbergite, a non-aluminous variety. The other constituent is light brown in color and not so clear. It constitutes a matrix of interpenetrating rods and feather-like aggregates. Throughout this aggregate the magnetite grains are scattered plentifully. Optical tests indicate that this mineral is also a pyroxene, and that it belongs to the colored monoclinic varieties included under augite. There is apparently no great difference between these two constituents in classification, but they have strikingly different development, which points to difference in chemical composition. The last mineral constitutes fully one-half of the slag. By reflected light very small matte particles can be discerned scattered throughout the mass. There are some rod-like crystals similar to those found in slag No. 9.

Slag No. 16.—This specimen is crystalline and made up of two constituents, one of which is pyroxene and the other magnetite. Very small areas are isotropic and are probably included glass. The prominent mineral occurs in long columnar growths, which in thin sections appear either as long crystals with irregular margins and very prominent longitudinal structure, due to multitudes of inclusions, or as a geometrically branching or graphic area which represents the same type of crystal cut cross-wise. The grains cut longitudinally have a green to yellowish green pleochroism, those cut cross-wise vary from green to yellowish brown. The mineral has a high extinction angle and double refraction, and the optical characteristics are those of ferriferous augite (or basaltic augite). This mineral constitutes about 75 per cent. of the slag. The rest is magnetite and undifferentiated residual glass, both of which occur as inclusions, but more especially as a sort of matrix

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which surrounds every crystal and fills the grooves or longitudinal re-entrant angles along their fluted sides.

On a critical analysis of the results set forth it is apparent that the most prominent constituent of the regular slags is a pyroxene, in some instances hedenbergite, in most cases augite. In one slag both are present. Olivine is also probably present and magnetite is a common constituent. These slags belong to the binary series, augite-olivine, in which, according to J. H. L. Vogt,[5] a eutectic is present, or they may be considered to belong to a ternary series, augite-olivine-magnetite. In the first-named series the eutectic composition, or the mixture of the two minerals that has the lowest melting-point, occurs when the silicate degree of the mixture is about 1.65. In the freezing of a slag, a member of this series, either augite or olivine will crystallize out first as the "excess substance," *i.e.*, that which is present above that required for the eutectic ratio, dependent upon whether the silicate degree of the slag be greater or less than 1.65. While the eutectic in metallic alloys consists of the conglomerate of the two entities which form, in a silicate, owing to viscosity, the eutectic usually congeals to a glass (the residual glass of the above description) and one member of the series, in this instance the olivine, may be largely in the glass and undistinguishable as a distinct mineral. The presence of an abundance of augite and practically no olivine shows these slags to belong on the augite side of the eutectic, and of a silicate degree greater than 1.65. This is not apparent from the chemical analysis, for calculations made in the ordinary manner, with all the iron present as ferrous oxide, give a silicate degree considerably lower; *e.g.*, the silicate degree of slag 15 is 1.22, which calls for olivine crystallization first.[6] This apparent discrepancy is readily explained when the presence of magnetite is considered. A very considerable portion of the iron shown by the analyses is in the form of magnetite. This has the effect of increasing the silicate degree of the slag very decidedly above that called for by the analysis. The source of this magnetite in the slags is not certain. It probably

[5] *Ueber die Schmelzpunkt-Erniedrigung der Silikatschmelzlösungen*, p. 114. Doelter, *Handbuch der Mineralchemie*, vol. i., p. 943 (1911).

[6] If 25 per cent. of the iron present in this slag is in the form of magnetite, the silicate degree becomes 1.45.

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comes from the converter-slag which is added in the furnaces, but part of it may be in the calcines of the charge, or be formed during smelting. The iron in the magnetite cannot be considered as flux for silica, and its presence is detrimental in so far as it crystallizes out of the slag at comparatively high temperatures, passing in part to the underlying matte, and when present in sufficient quantity making a mushy slag, which may entangle considerable matte and thus cause high-copper slags. The value of converter-slag as a flux seems questionable in many instances on account of the presence of this magnetite. Magnetite is present in many slags high in iron.

It is customary to consider the iron in slags to be present in the ferrous condition entirely, and combined with silica. That this is not the case can readily be seen from the results of the analyses in Table III., most of which do not add to 100 per cent. when the iron is calculated as ferrous oxide.[7] This, of course, is confirmed by the microscopic analysis. It has been noted frequently by different observers that many slags containing considerable iron are magnetic. It is also worthy of note that slags containing augite very likely contain ferric iron as well as ferrous iron, since both forms of the oxide enter into the constitution of the augite molecule. That this is the case with the reverberatory-slags under discussion is indicated by the brownish color of some of the augite found. The amount of magnetite found in the slags was sometimes very appreciable, more than 10 per cent. of the weight of the slag, thus removing a large portion of the iron from the influence of silica. This description gives some idea of the complex condition of the iron in some slags, and the effect it has on the nature of the slag.

III. Melting-Points of Slags.

In the discussion of certain thermal properties of slags such terms as "formation-temperature," melting-point, freezing-point, viscosity, superheating, and fluidity are frequently used, sometimes in a confusing manner, so that it seems

[7] Slag No. 15 contains 33.38 per cent. of iron. If 25 per cent. of this be calculated as Fe_3O_4, and 75 per cent. as FeO, the total of the slag-constituents will be increased by 0.79 per cent. only.

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desirable to view some of these properties in the light of recent investigation. From the metallurgical standpoint the most important physical property of a slag is the temperature at which it will readily flow from the furnace and permit the separation of metal and matte from it by gravity. This property of fluidity is closely related to the melting-point and to the viscosity (resistance to flow) in a certain temperature-range above the melting-point. The melting-temperature or point, defined as that temperature at which the solid slag passes to the liquid condition, is perhaps the property best suited for the purpose of investigation, as will be evident from what follows. The melting-point of a silicate, however, presents different aspects than the melting-point of metals or substances which are excellent conductors of heat and of low viscosity. When a mass of metal to which heat is being supplied from an exterior source at a sufficient rate begins to melt on the outside layers, the conduction of heat to the interior of the mass is so rapid that all of it practically reaches the melting-point at the same time. This is not true of silicates. In this case the melting of the mass takes an appreciable time and the time-temperature curve during the melting-stage is therefore an oblique one, and not the sharp, nearly horizontal jog that shows in the freezing-curve or melting-curve of metals. This time-lag in the melting or freezing process is most pronounced the more viscous the substance is at or near the melting-point and is very prominent in such substances as orthoclase and albite, or other silicates containing considerable alumina, and in some borates, but is not very pronounced in the iron, lime, or manganese silicates. Many silicates also possess the property of very appreciably undercooling below the true freezing-point before solidification or crystallization sets in and the latent heat of fusion is given out. Therefore in taking a freezing-point this almost invariably comes too low, sometimes by 100° or more. Moreover, the amount of lowering is not constant and may vary with each determination of the freezing-point. I spent several months fruitlessly in endeavoring to work out the slag series which follow by the freezing-point method. The data in Table IV. and in Fig. 1 illustrate these facts.

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TABLE IV.—*Melting- and Freezing-Curves on Slag No.* 16.

Melting-Curves.						Cooling-Curves.			
Mic. Volts.	Dif.	Mic. Volts.	Dif.	Mic. Volts.	Dif.	Mic. Volts.	Dif.	Mic. Volts.	Dif.
7,000	90	6,730	140	7,730	170	10,910	210	9,600	200
7,090	90	6,870	140	7,900	160	10,700	240	9,400	300
7,180	90	7,010	140	8,060	160	10,460	240	9,100	200
7,270	90	7,150	130	8,220	150	10,220	220	8,900	200
7,360	80	7,280	130	8,370	120	10,000	230	8,700	150
7,440	80	7,410	130	8,490	110	9,770	210	8,550	150
7,520	80	7,540	130	8,600	95	9,560	210	8,400	200
7,600	80	7,670	120	8,695	80	9,350	190	8,200	200
7,680	90	7,790	115	8,775	75	9,160	180	8,000	150
7,770	80	7,905	115	8,850	60	8,980	180	7,850	.....
7,850	80	8,020	110	8,910	50	8,800	.....	.....	.....
7,930	90	8,130	100	8,960	60				
8,020	80	8,230	100	9,020	55				
8,100	80	8,330	100	9,075	55				
8,180	90	8,430	80	9,130	70				
8,270	90	8,510	90	9,200	70				
8,360	70	8,600	80	9,270	80				
8,430	70	8,680	70	9,350	100				
8,500	70	8,750	60	9,450	110				
8,570	70	8,810	50	9,560	140				
8,640	60	8,860	50	9,700	230				
8,700	60	8,910	50	9,930	120				
8,760	50	8,960	50	10,050	110				
8,810	40	9,010	50	10,160	95				
8,850	50	9,060	40	10,255	85				
8,900	40	9,100	50	10,340	75				
8,940	40	9,150	55	10,415	65				
8,980	50	9,205	55	10,480	.....				
9,030	40	9,260	70	.....	.....				
9,070	40	9,330	90						
9,110	40	9,420	85	c. j.—28° C.					
9,150	50	9,505	70	.....	.....				
9,200	60	9,575	95	.....	.....				
9,260	60	9,670	140	.....	.....				
9,320	80	9,810	170	.....	.....				
9,400	80	9,980	135	.....	.....				
9,480	90	10,115	95	.....	.....				
9,570	90	10,210	90	.....	.....				
9,660	90	10,300	80	.....	.....				
9,750	160	10,380	70	.....	.....				
9,910	130	10,450	.....	.....	.....				
10,040	.....	.....	.....	.....	.....				
c. j.—22° C.		c. j.—28° C.							

Melting-points:

No. I. = 1,036° C.
No. II. = 1,036° C.
No. III. = 1,037° C.

NOTE.—Readings at minute intervals.

On slag No. 16, which is very nearly a bisilicate, only the melting-curve gives results. This slag is so viscous near its freezing-point that practically no inflection is to be noted in the cooling-curve. Somewhat better results are obtained with the cooling-curve if the rate of cooling is less than shown in

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the table, but they are far from conclusive. It may therefore be stated that the investigation of the freezing-points of slags is much less satisfactory than that of the melting-points, due to the great undercooling which takes place.

While undercooling[8] is a prominent phenomenon in silicates, overheating—viz., heating beyond the temperature of the true melting-point without liquefaction—is also possible, according to Doelter,[9] for many silicates, and, according to Day and Allen,[10] for certain very viscous materials, such as albite, orthoclase, or quartz. According to W. P. White,[11] it is negligible for many silicates, especially for those of the types under consideration.

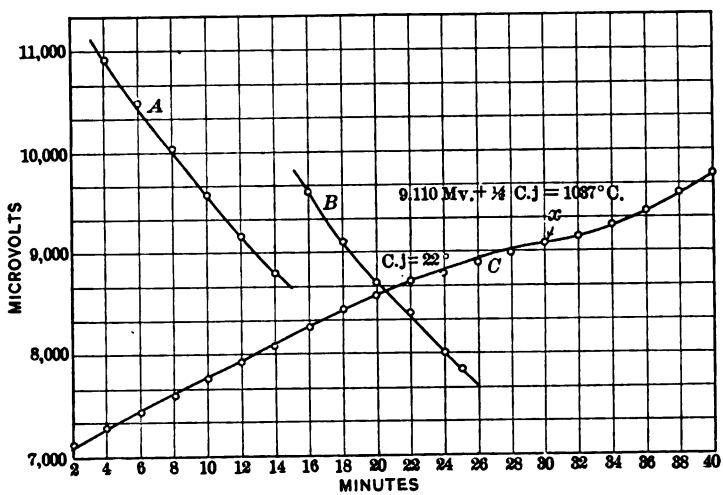

FIG. 1.—FREEZING- AND MELTING-CURVES OF SLAG NO. 16, ELECTRIC FURNACE.

The experimental evidence of this fact rests in being able to get the same melting-point repeatedly with different rates of heating. If overheating can take place, the melting-point comes higher with the increased heating-rate. In the work under consideration rather widely different heating-rates gave but very little change in the melting-point.

[8] W. P. White, Melting-Point Determination, *American Journal of Science*, Fourth Series, vol. xxviii., No. 167, p. 455 (Nov., 1909). C. Doelter, *Handbuch der Mineralchemie*, vol. i., p. 677 (1911).

[9] *Handbuch der Mineralchemie*, vol. i., p. 631 (1911).

[10] The Isomorphism and Thermal Properties of the Feldspars, *American Journal of Science*, Fourth Series, vol. xix., No. 110, p. 119 (Feb., 1905).

[11] *Ibid.*

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The term "formation-temperature" of a slag is usually taken to mean the temperature at which the mixture of constituents that go to form the slag becomes fluid. When closely considered this is a vague conception, for this temperature is a variable, depending on a number of factors, viz., rapidity of heating, the possible formation of a eutectic, size of particles of the mixture. It is also possible that in the mechanical mixture one substance may go partly into solution in the others, below the melting-point of the mass as a whole. The formation-point, considered as the temperature at which chemical combination takes place, may lie below the melting-point, as has been shown by Day and Shepherd for calcium silicates.[12] From the metallurgical standpoint, and with coarse material, the formation-temperature, so-called, lies above the melting-point, but it lacks definiteness for the purpose of investigation.[13] The upper melting-point of the formed slag (viz., not that of the eutectic, if such be present, but that at which the whole mass passes to the fluid state) is a definite and fixed point with those silicates in which overheating is not a phenomenon, and for that reason was adopted as the basis of this investigation. The viscosity of the slags under consideration increases with the silica- and alumina-contents and varies much for the different slags. For this reason it is necessary, aside from the melting-point, to determine what temperature is required to cause the slags to be sufficiently limpid to flow, *i.e.*, what amount of superheating is required above the melting-point. This degree of fluidity was determined by stirring with a heavy platinum rod and noting the temperature reading at the time of stirring.

The method of taking melting-points used in this investigation is based on the researches of Walter P. White,[14] to which reference should be made for a complete discussion of the subject. Briefly stated, the essential data are as follow: (1) A furnace is necessary in which the heating-rate is under perfect control. A properly-designed electric furnace is best for the purpose.

[12] *American Journal of Science*, Fourth Series, vol. xxii., No. 130, p. 268 (Oct., 1906).

[13] *Principles of Metallurgy*, p. 272 (1910). *Handbuch der Mineralchemie*, vol. i., pp. 646, 954 (1911). H. O. Hofman, *Mineral Industry*, vol. xvii., p. 597 (1908).

[14] Melting Point Determination; Melting Point Methods at High Temperatures, *American Journal of Science*, Fourth Series, vol. xxviii., No. 167, pp. 453 to 489 (Nov., 1909).

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(2) Accurately standardized thermo-couples are required. The hot-junction should be properly centered in the mass, the melting-point of which is to be determined. (3) It is essential to have means of accurately measuring small differences of potential generated in the thermo-couples. This is best done by means of a potentiometer and a delicate reflecting galvanometer.

The hot-junction of the thermo-couple is centered in the molten substance contained in a platinum crucible, being placed two-thirds down from the top of the molten mass. The contents of the crucible are then allowed to solidify within the furnace, care being taken to have the crucible so placed that it will heat uniformly from the walls of the furnace when the crucible is reheated. The particular position in the furnace is best determined by experiment. The thermo-couple always indicates the temperature of a small mass of material immediately surrounding the hot-junction of the couple.

Two methods of taking heating-curves are available. The current of the furnace is so adjusted and varied that the amount of heat supplied to the mass to be melted is constant. If the current on the furnace is kept constant, which means essentially that the heating-rate of the furnace will be constant, then the amount of heat supplied to the melting mass will be variable, for the amount of heat which passes to the mass from the furnace-walls depends upon their difference in temperature. This difference determines the heating-rate. At the beginning of the heating there is a certain difference in temperature between the furnace-wall and the crucible contents. When melting commences heat is absorbed by the crucible contents and its temperature-rise diminishes and falls behind that of the furnace-wall. Hence the temperature difference between furnace and crucible increases as the melting proceeds, and the rate of heat-supply is rapidly increased towards the end of the melting, with the result that this portion is greatly hurried, the melting-curve becoming oblique, and the rise or break in the curve after the melting of the material around the junction of the couple is not as sharp as it would be if the rate of heat-supply had been less. In fact, with too high a heating-rate the break may be much obscured. For examples of heating-curves with a variable heat-supply, in the manner just discussed, but with a comparatively low rate, consult Fig. 2, that of copper,

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and Fig. 3, that of salt. During the melting of the mass immediately surrounding the hot-junction of the thermo-couple the heating-rate is practically *nil* in substances that are good

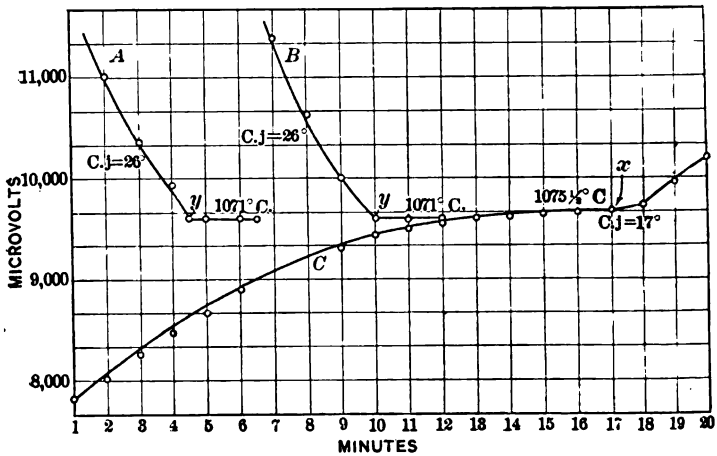

FIG. 2.—FREEZING- AND MELTING-CURVES OF COPPER.
Freezing-Points in Gas-Furnace; Melting-Point in Electric Furnace; Copper at 1,084° C.

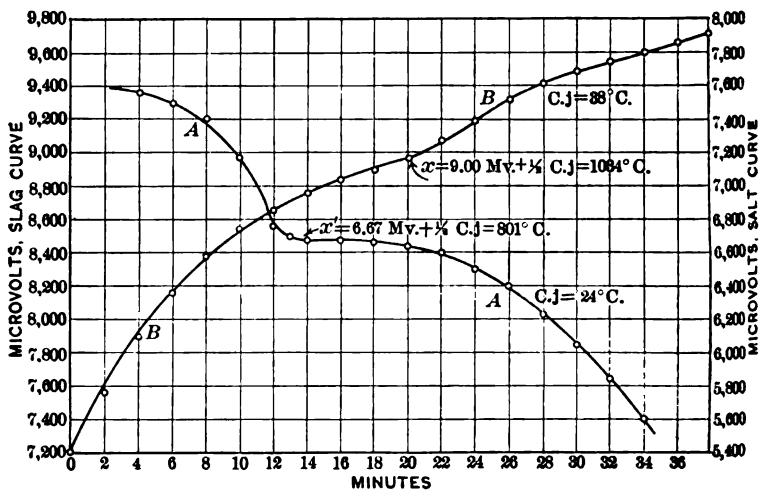

FIG. 3.—*A* = MELTING-CURVE OF SALT. *B* = MELTING CURVE OF SLAG No. 3, SERIES III.

conductors and not viscous, but in substances like silicates the rate decreases to a minimum, to be followed by a rise, as in the melting-curve of slag No. 3 in Fig. 3. The more rapid rise suc-

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ceeding the initial rise is probably due to convection-currents within the molten mass. The method of heating by variable heat-supply distorts the heating-curve as shown, and while by this method the upper melting-point can be determined within a few degrees if the heating-rate be properly adjusted, other melting-points, such as that of the eutectic, are obscured. Greater accuracy is obtained if the furnace is operated to maintain a constant difference of temperature between the furnace-wall and the melting mass at all stages of the melting. This can be done by means of a second or control couple placed in the furnace-chamber next the crucible at about the same level. The difference in reading between this couple and that in the melting mass is kept constant, or nearly so, by controlling the current supplied to the furnace by means of a rheostat. The resulting curve is much less oblique than that taken by the first method and more sharply defines the rise at the end of the melting, thus placing the melting-point more accurately, and also serves to give an accurate melting-curve, on which eutectic or residual meltings can be determined.

If the heating-rate be so adjusted, with a constant furnace-temperature, as not to be too high during the latter part of the melting interval, a curve results on which the break due to the melting of the mass surrounding the junction of the couple is sharp and pronounced and the melting-point determined within a degree or two. As the upper melting-points only were sought for in this work this method was adopted as sufficiently accurate. The same point may be obtained repeatedly, except when the heating-rate is too high. Then it will come a few degrees above the true point. Table III. gives three heating-curves on slag No. 16, the last two of which are taken with somewhat too high a rate; still, the displacement in the melting-point is small. Curve No. 1 of this slag is plotted in Fig. 1. In Fig. 3, *B* shows the heating-curve of slag No. 3 in series III. Heating-curves of slags in series I. and II. are shown in Figs. 4 and 5, respectively. The records of the heating show the melting-point better than the plotted curves. The melting-point is taken as lying between the reading recording the lowest rate and that recording the first increase thereafter. Very often a sharp deflection of the beam of light on the galvanometer scale is observed when the mass surrounding the junction

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melts. Thereafter the rise in rate may be either comparatively slow at first and then increase suddenly, or it may increase rapidly at once, dependent on the viscosity of the molten slag.

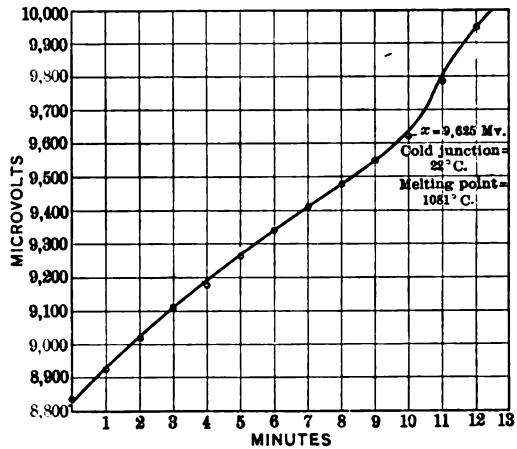

FIG. 4.—MELTING-CURVE OF SLAG No. 4, SERIES I.

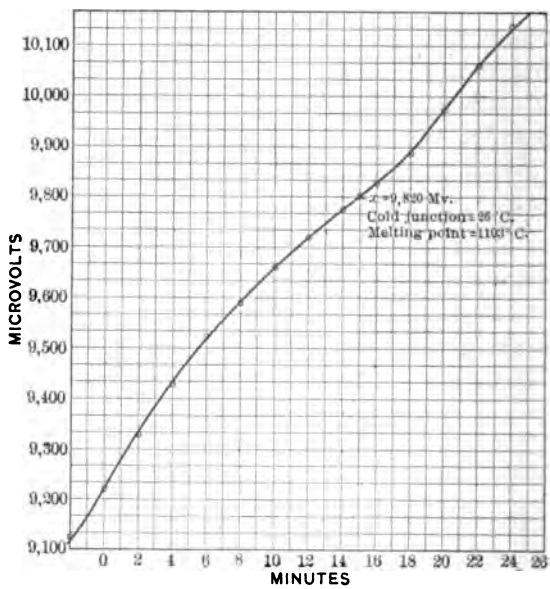

FIG. 5.—MELTING-CURVE OF SLAG No. 8, SERIES II.

If viscous, the rise is comparatively slow; if limpid, the rise is rapid. This leads to the conclusion that the rapid rise in rate is due to the equalization of temperature brought about by

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convection-currents, which are delayed in the viscous material until it becomes sufficiently superheated.

The Furnace.—The most suitable furnace for obtaining uniform heating-conditions is some form of electric-resistance furnace. In this work it was expected that temperatures above 1,400° C. would be required. Most furnaces on the market, with the exception of platinum-wound furnaces, are not available, as this temperature is beyond that for which they are

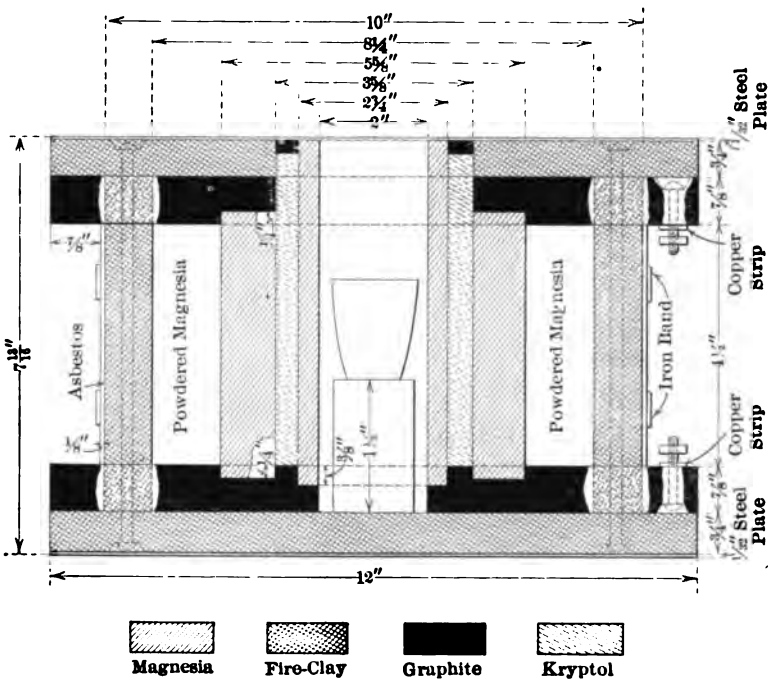

FIG. 6.—ELECTRIC-RESISTANCE FURNACE.

constructed. This refers to furnaces in which the heating-rate is under such control as to permit their use for scientific investigation. A foreign carbon-resistance furnace was imported, but proved unsuitable in the work. After considerable experimental work the furnace shown in detail drawing in Fig. 6 and in perspective in Fig. 7 was built, and gave most excellent service. The resistor is kryptol. The cross-section and length of the resistor cylinder are designed for a 115-volt direct current. The current-consumption is from 10 to 30 amperes,

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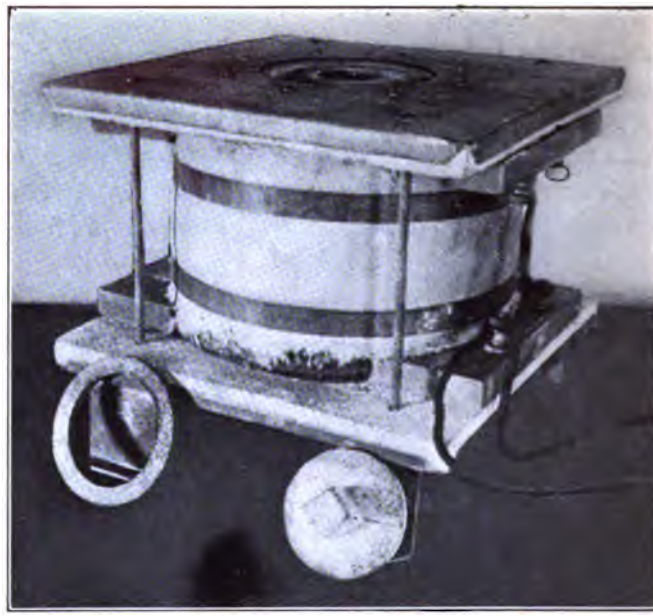

FIG. 7.—ELECTRIC FURNACE. KRYPTOL.

FIG. 8.—ELECTRIC FURNACE. PLATINUM WIRE.

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dependent upon the temperature to be obtained. With the latter current the limit of the platinum-rhodium thermo-couple is reached in about 4 hr., if the furnace is protected against excessive radiation. The essential features[15] of the furnace are the following: (1) Two concentric cylindrical tubes, the inner one of which forms the heating-chamber, made of either magnesia (calcined magnesite furnished by the Harbison-Walker Co., Pittsburg) or alundum, and the cylindrical space between which, when filled with a resistor like kryptol, becomes the resistor cylinder. (2) Uniform and continuous contact of the resistor on its whole cross-section with the lower graphite electrode plate. (3) A resistor of uniform cross-section throughout its length. (4) The positive and firm insertion of the tubes in the electrode plates, thus maintaining the resistor chamber in perfect shape. (5) That any part can be readily and cheaply renewed. (6) Simplicity and compactness of construction, preventing racking and distortion during heating.

A practical difficulty experienced with most furnaces using a granular carbon resistor is the uneven heating of the furnace space, usually due to poor design of the shape of the resistor space. The furnace described gives very little trouble in this respect, and then only when, after considerable use with high currents, arcing takes place locally in the resistor, destroying the kryptol and fusing small portions of the tubes. An error in the design of many resistance-furnaces seems to be in the effort made to operate with too small an amount of current, making the cross-section of the resistor, either granular or wire, so small that local overheating, with consequent burning-out, readily takes place. This saving of a few watts of current is more than made up in expense for repairs and annoyance.

A furnace of the type used by Day and Allen[16] was also used for part of the work. It consists of a magnesia tube 6.5 in. high and 2.25 in. inside diameter, on the inside of which in a spiral groove are wound 25 coils of platinum wire, No. 18 B & S. gauge. This wire is 17 ft. long and 0.0403 in. in diameter and weighs 91.2 g. The furnace was designed for a maxi-

[15] Patent applied for by C. H. Fulton and W. A. Coursen.

[16] Temperature Measurements to 1600° C., *Physical Review*, vol. xix., No. 3, p. 177 (Sept., 1904). *American Journal of Science*, Fourth Series, vol. xxvi., No. 155, p. 411 (Nov., 1908); vol. xxviii., No. 167, p. 474 (Nov., 1909).

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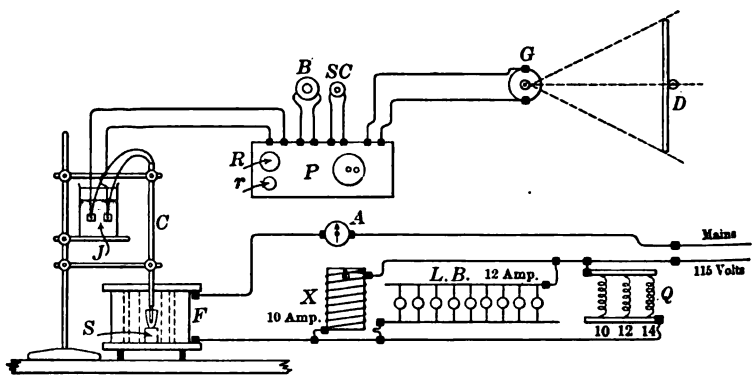
mum current of 45 amperes at a voltage of 115. For the work in question a current of from 10 to 20 amperes was used. The tube is inclosed in fire-clay blocks, as shown in Fig. 8. This particular model was designed to permit of pouring the contents of crucibles within the furnace directly into a calorimeter, after opening a slide door which closes the bottom of the heating-tube. By this method specific heats of slag were determined. This furnace is eminently satisfactory, but the cost, at the present price of platinum, is very high.



F, Furnace.
J, Cold-junction.
S, Crucible.
C, Thermo-couple tube.
P, Potentiometer.
B, Storage-battery.
A, Ammeter.
SC, Standard cell.
G, Reflecting galvanometer.
D, Lamp and scale.
X, Adjustable resistance.
L.B. Lamp resistance.
Q, Rheostat.

FIG. 9.—SET-UP FOR TAKING MELTING-POINTS.

Fig. 9 gives a diagrammatic sketch of the set-up required for the taking of melting-points.[17] Two types of potentiometers were used: one a thermo-couple potentiometer made by the Cambridge Scientific Instrument Co., and the other a precision thermo-couple potentiometer made by the Leeds-Northrup Co. The last instrument measures direct to 0.5 microvolt. The reflecting galvanometer was either a Broca instrument, made by the Cambridge Scientific Instrument Co., or a Leeds-North-

[17] For a description of the potentiometer method of measuring temperatures consult *Principles of Metallurgy*, p. 165 (1910).

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rup D'Arsonval instrument, type HS. Either instrument is very sensitive to small differences of current. In taking readings the beam of light was brought to zero on the scale at the end of the time-interval and the microvolt reading taken on the potentiometer. The current on the furnace was kept constant by means of the rheostat, which has an adjustable resistance.

IV. Three Series of Slags.

Three series of slag were investigated. In many smelting-problems the iron, silica, and alumina bear practically a constant ratio to each other in the smelting-mixture, for with sulphur and copper these substances make up the bulk of the ores, and the only variable in the smelting-mixture is lime, added as limestone flux. On this assumption three basal or ratio slags were selected, in each of which there is a different ratio of SiO_2 : FeO : Al_2O_3, and to which were added different amounts of lime to make the slags of a given series.

Table V.—*Composition of Basal Slags.*

Number of Series.	Composition.				Ratio of SiO_2 : FeO : Al_2O_3.
	SiO_2.	FeO.	Al_2O_3.	CaO.	
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	
I.	42.65	36.12	11.30	6.20	377 : 319 : 100
II.	44.50	38.20	11.85	5.10	376 : 323 : 100
III.	44.30	43.1	4.85	6.80	914 : 888 : 100

Two of the series are relatively high in alumina and one is low in alumina. For a given silicate degree, series I. is lowest in iron as a base and series III. is highest. The slags of the three series therefore furnish data showing the effect of varying silicate degree, and the effect of iron and lime as a base, and also the effect of alumina, within certain limits. The materials for the slags of series I. and II. were the ores and flux of the actual smelting-mixture. These were taken to conform to the actual conditions obtaining at the plant for which a part of this investigation was carried out. Table VI. gives the composition of carefully prepared samples, which were used in making the mixture.

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TABLE VI.—*Composition of Material.*

Material.	Composition.								
	SiO_2.	Fe as Fe_2O_3.	Fe.	Al_2O_3.	CaO.	MgO.	Cu.	Loss.	Total.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent	Per Cent.	Per Cent.	Per Cent.
Limestone.....	1.94	1.09	.....	0.58	52.54	1.09	.....	42.49	99.73
Converter-slag,	17.80	73.33	51.29	4.85	1.16	0.47	6.18	.....	.....
Calcines.....	29.10	30.24	.....	5.80	5.44	0.48	16.98	.....	.....
Tank-slimes.....	51.32	14.08	.....	11.24	1.12	.....	6.88	.....	.....

In making the basal slags of the first two series, the above substances were mixed in the proper proportions to yield from 75 to 100 g. of slag of approximately the desired composition. This mixture was carefully smelted down in the electric furnace, in cylindrical crucibles turned from pure Acheson graphite. The fusion was held at a comparatively high temperature for some time, but not high enough to cause any serious "boiling" of the matte, and then allowed to solidify in the furnace. When cold the mass readily drops from the crucible and the matte can be removed from the slag in the form of a button. The slag was then resmelted in the same crucible at a somewhat higher temperature to give opportunity for a second settling of matte, although usually only isolated globules were found after the second smelting. The slag was then crushed and about 20 g. ground fine in an agate mortar, and the metallic iron reduced by the action of the graphite crucible was removed by a magnet. The equivalent of this iron in ferrous oxalate was returned to the slag. This mixture was then placed in a weighed platinum crucible 1.25 in. deep and $\frac{5}{8}$ in. in diameter, viz., a narrow high crucible, and slowly melted down into a homogeneous slag. After cooling, the weight of the slag was taken. The melting-point of the slag was then determined by means of the bare thermo-couple, *i.e.*, no protective tube was used. The insulating-tubes reached to within about 0.75 in. of the hot-junction. After the last determination the slag was remelted, the couple removed and the slag allowed to solidify. Then limestone was added either in 0.5-, 1- or 2-g. additions, the crucible replaced in the furnace and the new slag melted down while stirring with a platinum rod. The temperature was then raised until the slag was thinly fluid, the thermo-couple care-

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fully inserted and centered, and after solidification reheated and new melting-points taken. In this manner the points of the whole series were taken consecutively. After the final determination the crucible with its contents was weighed, the slag then remelted until thinly fluid and then poured into water in a porcelain dish to get the sample for the chemical analysis. From the weights of the slag, the limestone taken, and the analysis, the composition of the slags of the series was calculated. This method was adopted because of its neatness and the little wear it caused on platinum, for a whole series could be run without the necessity of cleaning slag from the crucible, except at the end. The possible error in the composition of each slag due to calculating back from the final analysis is probably very small with the weighings carefully made, and with no dusting on the addition of limestone.

For the slags of series III., pure analyzed reagents, alumina (ignited), silica, calcium carbonate, and ferrous oxalate were used to make the basal slag. Otherwise the procedure for the series was the same as for the first two, except that the basal slag was analyzed and the other slags of the series caculated from this analysis and the original weight and the limestone addition. Tables VII., VIII., and IX. give the data on the three series in condensed form.

TABLE VII.—*Composition and Melting-Point of Slags of Series I.*

Number.	Composition.					Silicate Degree.	Melting-Point.		
	SiO_2.	FeO.	CaO.	Al_2O_3.	Total.		Millivolt Readings.	⅓ Cold-Junction.	Melting-Point.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.			°C.	°C.
1.	42.65	36.12	6.20	11.30	96.37	1.40	8.550	9	979
2.	40.66	34.33	10.80	10.70	96.49	1.28	8.815	16	1,011
3.	38.80	32.60	14.95	10.02	96.37	1.19	9.380	10	1,060
4.	36.90	31.10	18.52	9.75	96.27	1.09	9.625	11	1,081
5.	35.40	29.65	22.12	9.35	96.52	1.02	10.250	12	1,137
6.	33.92	28.48	25.14	9.00	96.54	0.96	10.420	14	1,151

Original weight of slag = 20 g. Weight of slag plus additions = 25.751 g.
Weight of final slag in crucible = 25.0575 g. Difference = 0.6935 g. on rod and couple.

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TABLE VIII.—*Composition and Melting-Point of Slags of Series II.*

Number.	Composition.					Silicate Degree.	Melting-Point.		
	SiO_2.	FeO.	CaO.	Al_2O_3.	Total.		Millivolt Readings.	½ Cold-Junction.	Melting-Point.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.			° C.	° C.
1.	44.50	38.20	5.10	11.85	99.65	1.51	9.260	13	1,050
2.	44.00	37.80	6.14	11.73	99.67	1.48	9.000	13	1,025
3.	43.50	37.30	7.15	11.60	99.55	1.45	8.600	13	988
4.	43.20	37.00	8.16	11.51	99.87	1.42	8.600	7	982
5.	42.25	36.20	10.04	11.23	99.72	1.38	8.850	10	1,010
6.	40.50	34.65	13.60	10.75	99.50	1.28	9.300	13	1,054
7.	38.90	33.25	16.80	10.40	99.35	1.20	9.700	14	1,091
8.	37.55	32.10	19.90	10.05	99.60	1.13	9.820	13	1,103
9.	36.20	30.807	22.56	9.67	99.30	1.06	9.840	15	1,107

Original weight of slag = 23.645 g. Weight of slag plus additions = 29.683 g. Weight of final slag in crucible = 28.891 g. Difference = 0.814 on rod and couple.

TABLE IX.—*Composition and Melting-Point of Slags of Series III.*

Number.	Composition.					Silicate Degree.	Melting-Point.		
	SiO_2.	FeO.	Al_2O_3.	CaO.	Total.		Millivolt Readings.	½ Cold-Junction.	Melting-Point.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.			° C.	° C.
1.	44.3	43.10	4.85	6.80	99.05	1.66	8.750	10	1,000
2.	42.65	41.51	4.69	10.29	99.14	1.54	8.850	10	1,010
3.	41.11	40.01	4.51	13.53	99.16	1.44	9.000	19	1,034
4.	38.34	37.22	4.21	19.35	99.12	1.27	9.740	15	1,097
5.	35.92	34.97	3.94	24.43	99.26	1.13	10.540	17	1,167
6.	33.79	32.89	3.71	28.92	99.31	1.03	11.000	22	1,207
7.	31.90	31.05	3.50	32.90	99.35	0.934	10.690	26	1,186

Original weight of slag = 14.40 g.

The melting-point curves constructed from the data of the three series are given in Figs. 10, 11, and 12. In Figs. 11 and 12 the lower curve is the melting-point curve and the upper one is the fluidity-curve. The curves are plotted on the basis of silicate degree and temperature, a given silicate degree representing a certain mixture of two minerals, one an ortho-silicate and the other a meta-silicate. The percentage of lime in the slags is also indicated. The preceding tables give the composition of the slags.

From the appearance of the curves, the series fall into a type

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in which a eutectic, or mixture of a lowest melting-point, occurs. While no petrographic examination was made of the slags of these series, they are of similar composition and type as the reverberatory-slags discussed in Section II. of the pres-

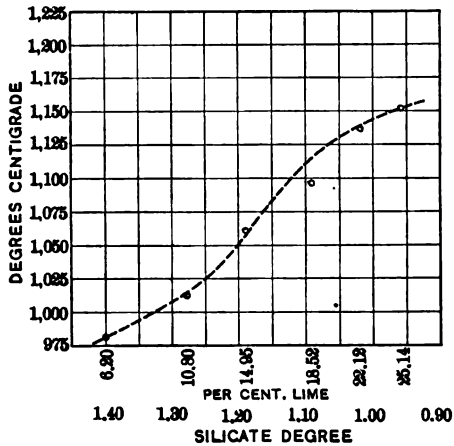

FIG. 10.—MELTING-POINT CURVE, FIRST SERIES.

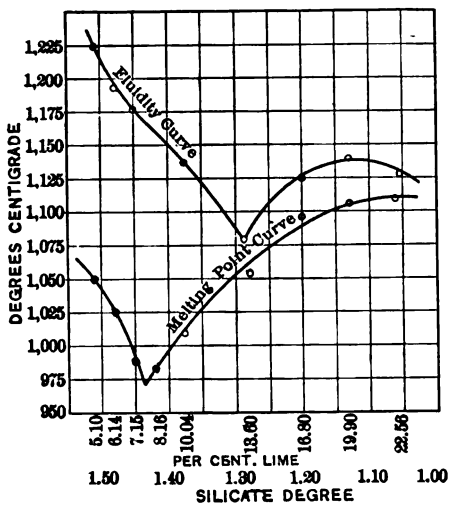

FIG. 11.—MELTING-POINT CURVE, FLUIDITY CURVE, SECOND SERIES.

ent paper. There seems but little doubt that series III. is an augite-olivine one, while series I. and II. are perhaps the same, but more probably augite-melilite, considering the amount of alumina present. The silicate degree of the eutectic of series

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III. falls at about 1.62, which is close to what J. H. L. Vogt states it to be for a certain augite-olivine series. The silicate degree of the eutectic of series II. falls at about 1.44, which corresponds to what it should be approximately for a certain augite-melilite series, according to the same authority. It will

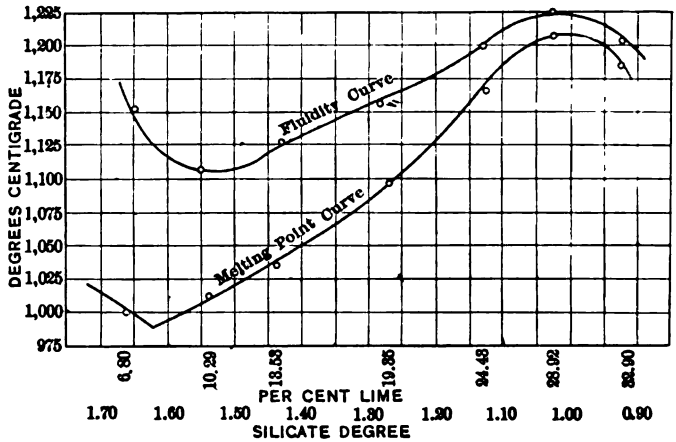

FIG. 12.—MELTING-POINT CURVE, FLUIDITY CURVE, THIRD SERIES.

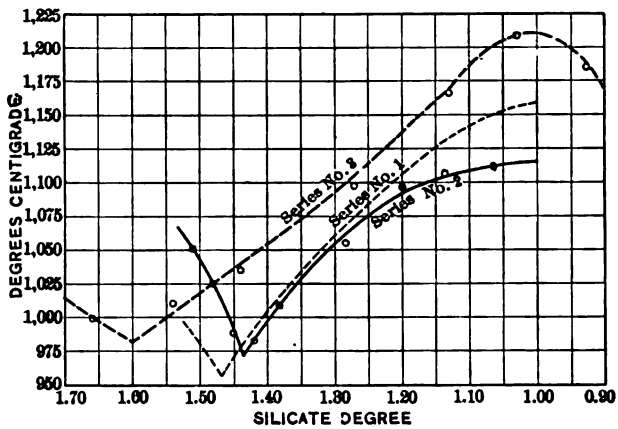

FIG. 13.—MELTING-POINT CURVES.

be noted that the total of the constituents of the slags in series I. adds up to but 96.5 per cent. It is probable that in these slags some magnetite is still present, which has the effect of increasing the silicate degree above that calculated. This will displace the curve to the left in Fig. 9. That this should be

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done is perhaps apparent when it is noted that no maximum is reached in this curve at a silicate degree of 1, as is done in series II. and III. A maximum is expected at this point in a series of this type, for here occurs one of the end-members of the series, viz., either olivine (lime-iron olivine) or melilite.

The other maximum occurs at a silicate degree of 2, or the augite end. It is also very probable that in series I., the first one to be worked out, the eutectic was missed, and is located between slags 1 and 2. That this is so is perhaps apparent when series II. is considered, which was worked out with much smaller addition of limestone in this region. On this assumption the melting-point curve for series I. is replotted in dotted lines in Fig. 13, with the melting-point curves of the other two series.

Discussion of Results.—From an inspection of the curves in Fig. 13 and the composition of the slags given in the preceding tables, it is apparent that the slags of lowest melting-points lie roughly between a silicate degree of 1.4 and 1.65, dependent upon the base composition. This confirms what has long been the opinion of copper-metallurgists. With a given silicate degree, within the range of 1.70 to 1.00, the replacement of iron and lime by some alumina, up to 11 per cent., lowers the melting-point, as is apparent when the curves of series I. and II. are compared with the curve of series III. In a given slag of certain silicate degree within the range specified, the replacement of iron by lime raises the melting-point somewhat. This, however, does not mean that a pure ferrous bisilicate, for instance, has a lower melting-point than one in which some iron is replaced by lime, for this condition would be expressed by the series hypersthene-augite, in which a eutectic may also occur. It has reference to series of minerals in each of which lime is necessarily present, as augite-melilite. Then of two slags each of the same silicate degree, the one with relatively more lime has the higher melting-point. The slags investigated have melting-points that lie between 970° and 1,220° C. The slags of higher silicate degree have comparatively low melting-points, and that portion of the curve to the left of the eutectic, towards the augite end, rises very gradually. For example, slag No. 16 (Table III.), augite with some magnetite, practically a bisilicate, has a melting-point of but 1,037° C. It is,

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however, probable that the presence of magnetite lowers the melting-point of this slag somewhat.

When, however, the fluidity-curves are examined in connection with the melting-point curves it will be noted that the gap between the two curves widens rapidly as the silicate degree increases, and while from 25° to 40° of superheating is sufficient in the vicinity of the ortho-silicate, from 150 to 175° is required to superheat to fluidity at a silicate degree of 1.5. By consulting Figs. 11 (high alumina) and 12 (low alumina) it may be seen that increased alumina accentuates the gap greatly, and the more highly aluminous slags, due to their great viscosity, require much superheating above the melting-point in order to obtain enough fluidity. It is this phenomenon which makes high-silica, high-alumina slags so difficult to work. As already noted, the minerals orthoclase and albite, high in alumina and silica, are extremely viscous.

Slags approaching near the bisilicate are looked upon with suspicion for reverberatory-furnace work particularly. Of late, however, the incentive to make the more highly siliceous slags has been great. On the score of formation-temperature and melting-point nothing can be urged against these slags, but they do require much superheating, especially if alumina rises above 5 or 6 per cent. There seems to be no good reason why they could not be made in furnaces of ample temperature reserve, such as the present oil-fired furnaces.

In conclusion, I desire to express my cordial acknowledgment for assistance rendered in various ways by Prof. M. F. Coolbaugh, W. A. Coursen, and Floyd Brooks. I am permitted to say that part of the work described was carried out for the Steptoe Valley Smelting & Mining Co. S. S. Sörensen kindly furnished most of the samples used, for which, and for suggestions made, I express my appreciation.

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The Development of the Reverberatory Furnace for Smelting Copper-Ores.*

BY E. P. MATHEWSON, ANACONDA, MONT.

(Cleveland Meeting, October, 1912.)

HISTORICAL.

THE early development of the reverberatory furnace for smelting copper-ores was the work of the Welsh smelters, particularly those of Swansea. The first record of a reverberatory furnace is made by Jars, who states that in 1755 copper-smelting was effected in reverberatory furnaces at Middleton-Tyas, in Yorkshire, England. The first patent, of any importance, for improvements in reverberatory furnaces was granted to Thomas Williams in 1778 for the granulation of the regulus. The next was that of William Evetts, in Sheffield, in 1812, for the cooling of the fire-bridge by the admission of air.

Charge-hoppers above the furnaces were in common use in 1848. The size of the hearth of the furnaces about the beginning of the 19th century was commonly 11 by 8 feet.

No material development was made until the Welsh process was brought to Colorado, where Richard Pearce, as manager of the Argo works, near Denver, developed the furnace to meet the requirements of a custom plant, under keen competition with lead-silver smelting-plants, using blast-furnaces.

His improvements are outlined in Fig. 1, elaborated by permission from E. D. Peters's excellent work,[1] to which the reader is referred for further details. This figure shows that the furnaces were in 1878, 9 ft. 8 in. by 15 ft. in hearth; and in 1894, 16 by 35 ft., while the capacity of the furnace had been increased from 12 tons per 24 hr. to 50 tons in the same period.

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* Presented also at the Joint Meeting of the Institute and Section IIIa, Metallurgy and Mining, of the Eighth International Congress of Applied Chemistry, New York, N. Y., September, 1912, and here published by mutual agreement.

[1] *Modern Copper Smelting.*

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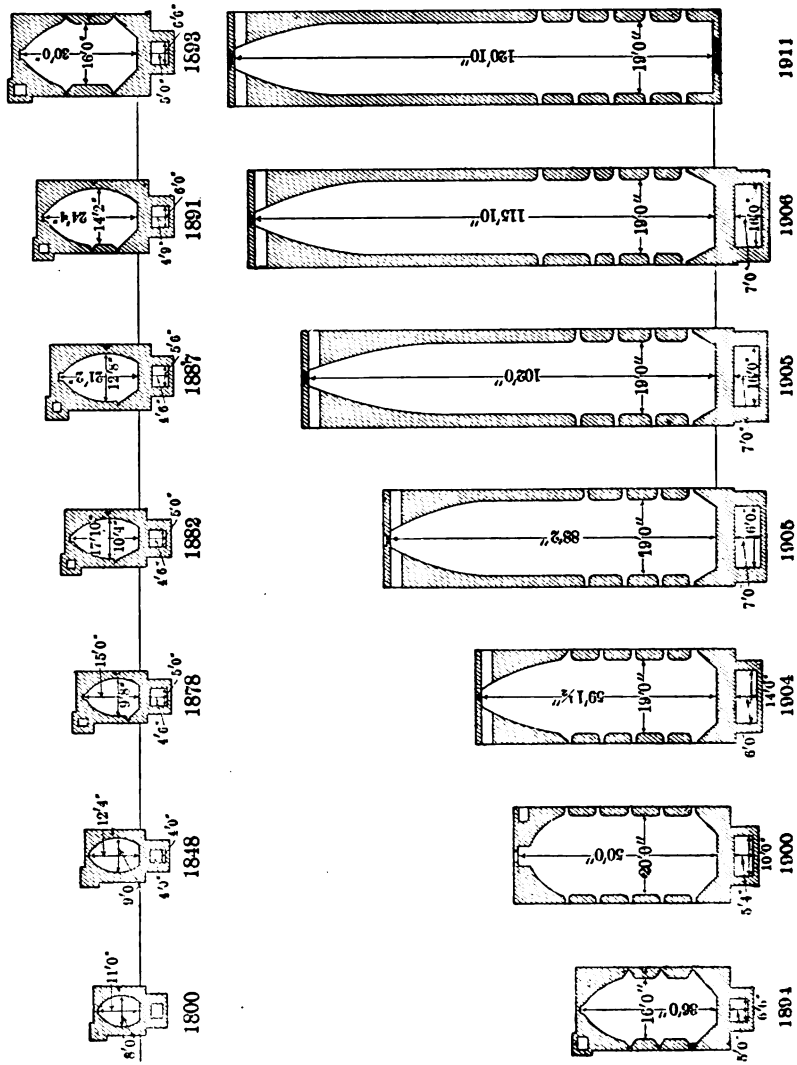

FIG. 1.—DEVELOPMENT IN SIZE OF THE REVERBERATORY FURNACE.
ELABORATED FROM PETERS'S *Modern Copper Smelting*.

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The next step in development was made in Butte, Mont., by the Colorado Smelting Co.—this plant being, at that time, affiliated with the Argo works, so that Mr. Pearce's influence was apparent. The step referred to was the lengthening of the hearth to 50 ft., with consequent increase in capacity to 105 tons in 24 hours.

The first furnace of this size—built from the Colorado Smelting Co.'s plans—was constructed at the Butte & Boston plant in Butte, Mont., in the year 1900. Details of Montana practice, up to the year 1902, will be found in Prof. H. O. Hofman's paper.[2]

The 50-ft. furnace became very popular in Butte and Anaconda, and when the Washoe smelter was built, in 1900–1902, the 50-ft. reverberatory was adopted and 14 of these furnaces were installed.

The next improvement was the addition of lime-rock to the charge of the calciners, so that it was thoroughly mixed into the charge before dropping the same into the reverberatories. This apparently insignificant change resulted in increasing the capacity of the furnace greatly.

The next step was the building of a furnace with a 60-ft. hearth, the results from which were so encouraging that a furnace with a hearth 85 ft. long was tried, with corresponding increase in tonnage. Then a radical step was taken at the Washoe plant, of connecting two 50-ft. furnaces, making a single furnace with a hearth 102 ft. long. The saving in fuel, and the increased tonnage therefrom, caused the management to build a furnace with a hearth 116 ft. long. The results of these experiments are given in the tabulated data below, compiled by William Wraith, Superintendent of the Washoe smelter.

Effect of Lengthening Reverberatory Furnaces at the Washoe Smelter, Anaconda, Mont.

The reverberatory furnace as originally built at the Washoe smelter had a hearth-area of 19 by 50 ft. After some months of operation it was decided that a longer furnace could be operated to a better advantage. To determine the length best suited to the conditions, one furnace was lengthened to 60 ft.;

[2] Notes on the Metallurgy of Copper of Montana, *Trans.*, xxxiv., 258 to 316 (1903).

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another to 85 ft.; another to 102 ft.; another to 112 ft.; and, finally, to 116 ft. in length.

The draft at the bridge-wall was from 0.75 to 1 in. of water; the fire-box area, 7 by 16 feet.

The coal used was from the Anaconda Copper Mining Co.'s mine at Diamondville, Wyo., having the following average proximate analysis and thermal values:

Analysis.	Per Cent.	Thermal Value. Wet Coal.	Dry Coal.
H_2O.....	6.13	11,710 B.t.u.	12,390 B.t.u.
Volatile matter.....	36.28		
Fixed carbon.....	45.42		
Ash.....	12.17		

There was some variation in the quality of the coal—ash ranging from 6 to 16 per cent., and the thermal value from 10,750 to 12,000 B.t.u. per lb. of wet coal; or, from 11,000 to 13,200 B.t.u. per lb. of dry coal.

There is also some variation in material smelted and in manipulation by the different furnace-crews.

Hearth-Areas of Furnaces.	Tonnage Per 24 Hours.	Ratios of Cupreous Material to Coal.	Averages.
Feet.	Tons.		
19 by 50	121.74	2.75	For 1 year.
19 by 60	190.7	3.94	For 7 months.
19 by 85	234.1	4.13	For 7 months.
19 by 102	264.9	4.31	For 4 years.
16 by 112	267.1	4.30	For 4 years.
19 by 116	270.1	4.19	For 4 years.

The copper-assays of the slags from the different reverberatory furnaces averaged:

	Copper. Per Cent.
Furnace No. 1, with 50-ft. hearth.. Sept., 1903–March, 1904.....	0.50
Furnace No. 6, with 60-ft. hearth.. March–Sept., 1904.....	0.44
Furnace No. 1, with 85-ft. hearth.. May–Dec., 1904.....	0.42
Furnace No. 1, with 102-ft. hearth.. Jan., 1906–May, 1911.....	0.39
Furnace No. 4, with 112-ft. hearth.. July, 1906–May, 1911.....	0.38
Furnace No. 1, with 116-ft. hearth.. Jan., 1906–May, 1911.....	0.36

Reverberatory furnaces in different localities present their own conditions, and the length of furnaces will be found to be established by the type of fuel, draft, tonnage to be smelted, and investment.

About this time, the Garfield plant of the American Smelters Securities Co. was being built near Salt Lake City, Utah,

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and for that plant furnaces with 100-ft. hearths were adopted. Then, at Humboldt, Ariz., the Arizona Commercial Copper Co. built two furnaces of 100-ft. hearth, using oil for fuel. The Steptoe plant, at McGill, Nev., followed with five furnaces with hearths 110 ft. long, coal-fired, changed to oil-firing in 1911. The Cananea Copper Co. put in one furnace with hearth 100 ft. long, for flue-dust, in 1906; followed by a second in 1911. Experiments were tried with coal-dust firing, but cheap oil being available, the coal-dust experiments were discontinued and oil substituted.

The reader is referred to Dr. Ricketts's article on experiments in reverberatory practice at Cananea.[3]

Table I. gives important data on the best modern installations of reverberatory furnaces, and Table II. presents the details of the chemical composition of the charge, the slags, the mattes, and the fuels. Fig. 3 is a diagram showing the tonnage- and assay-curves of the Anaconda furnace.

The features that distinguish the modern reverberatory are: its length; solid bottom (monolith of quartz fused); structural steel conker-plates for bridge-wall; frequent charging; infrequent skimming of slag; the large body of molten matte retained in the furnace to assist in melting and distributing the charge; no leveling of charge by means of rabble; thick roof (from 15 to 20 in. of silica brick); fettling only once a month in many plants, practically continuous operation; recovery of waste heat in the form of steam; and, where coal is used for fuel, the recovery of the unburnt coal from the ashes.

During the development of the reverberatory furnace to its present state of efficiency, a great many schemes were tried out and abandoned; for example, the preheating of the air, by passing it under the furnace-bottom or around the walls of the flue or fire-box, and forced blast under the grate.

A great variety of fuels has been used, and is being used, in reverberatory practice; for example, at Kyshtim, Perm Government, Russia, producer-gas from wood is in use, with very gratifying results. At this plant the hearth of the furnace is about 35 ft. between ports and 15 ft. wide; the gas is made from pine wood, and the moisture and tars are scrubbed from the gas before going to the furnace.

[3] *Transactions of the Institution of Mining and Metallurgy*, vol. xix., p. 147 (1909–10).

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TABLE I.—*Details of Modern Reverberatory Furnace-Smelting in America.*

	Anaconda. (Montana.)	Cananea. (Mexico.)	Great Falls. (Montana.)	Garfield. (Utah.)	Steptoe (McGill). (Nevada.)		Tooele. (Utah.)
					Coal-Fired.	Oil-Fired.	
Number of furnaces	8	2	3	6	5		5
Average tonnage per day	253— (229.5 T. M.)	192.3 (174.4 T. M.)	200— (181.4 T. M.)	240— (217.7 T. M.)	239— (216.8 T. M.)	322— (292.1 T. M.)	225— (204.1 T. M.)
Average weight of charge	15 tons. (13.6 T. M.)	7.5 to 9 tons. (6.8–8.2 T. M.)	35 tons. (31.75 T. M.)	6 to 24 tons. (5.44–21.77 T. M.)	14 tons (var.) (12.7 T. M.)	14 tons (var.) (12.7 T. M.)	13 tons. (11.8 T. M.)
Average tons charge per ton fuel	4.25	5.86	2.00	3.50 to 4.00	3.24	5.80	3.80
Average temperature of charge entering furnace	950° F. (Est.) 510° C.)	500° to 550° F. (200°–288° C.)	788° F. (420° C.)	400° to 700° F. (204°–371° C.)	500° F. approx. (260° C.)	500° F. (250° C.)	750° F. (390° C.)
Kind of fuel used	Run of mine coal.	California crude oil.	Producer gas	Bituminous coal.	.....	California crude oil	Run of mine coal.
Character of charge:							
Calcine (hot), per cent	84.95	50.0	66.00	75 to 78	61.3	56.8	95.00
Calcine (cold), per cent	.....	.....	.....	19 to 16	.....	.....	.....
Flue-dust, per cent	15.05	50.00	34.00	6.00	1.2	1.4	5.00
Conv.-slag, per cent	.....	.....	.....	.....	9 8	7.1	.....
Secondaries (cold), per cent	.....	.....	.....	.....	12.5	15.2	.....
Flux and fettling-ores (cold and wet), per cent.	.....	.....	.....	.....	15 2	19.5	.....
Dimensions of hearth	111′–8″ x 19′–0″ (31 04 x 5.79 m.)	100″ x 19′ (30.48 x 5.79 m.)	41′–6″ x 15′–9″ (12.65 x 4.80 m.)	112′ x 19′ (34.14 x 5.79 m.)	(120′–10″ x 19′.) (36.83 x 5.79 m.)		102′ x 19′ (31.09 x 5.79 m.)
Dimensions of fire-box	7′ x 16′ (2.13 x 4.88 m.)	None.	Gas-fired.	7′ x 18′ (2.13 x 5.49 m.)	None.		7′ x 16′ (2.13 x 4.88 m.)
Top of grate-bars to top of bridge	24″ (61 cm.)	None.	Gas-fired.	33″ (84 cm.)	33–1-4″ (84 cm.)		40″ (102 cm.)
Top of grate-bars to under side of roof	70″ (178 cm.)	None.	Gas-fired.	45″ to 48″ (114–122 cm.)	85–3-8″ (217 cm.)		78″ (198 cm.)
Height of "verb" or Vulcatory above skim-plate	36″ (91 cm.)	36–1-2″ (93 cm.)	Gas-fired.	36″ to 38″ (91–97 cm.)	36″ (91 cm.)		36″ (91 cm.)
Dimensions at throat of furnace	7′ x 4′–11″ (2.13 x 1.50 m.)	7′ x 1′–9″ (2.13 x 0.53 m.)	Gas-fired.	7′ x 3′–3″ (2.13 x 0.99 m.)	6′–10″ x 3′ (2.08 x 0.91 m.)		7′ x 4′–1″ (2.13 x 1.24 m.)
Dimensions of flue beyond throat (damper-flue)	36″ by 54″ (91 x 137 cm.)	7′ x 6′ (213 x 183 cm.)	....	89″ x 64″ (99 x 163 cm.)	(Not given.)		36″ x 60″ (91 x 152 cm.)
Draft at bridge in inches of water	(natural draft.) 0.75–1 (1.9–2 54 cm.)	(No data)	0.2 (0.5 cm.)	0.6 (1.5 cm.)	0.2–0.5 (0.5 cm–1.3 cm.)		0.8 (2 cm.)
Draft at throat in inches of water	1.3–1.7 (3 8–4.8 cm.)	0 13 (0.33 cm.)	0 3 (0.76 cm.)	0.8 (2 cm)	0.7–1.2 (1.78–3.05 cm.)	0.9 (2.29 cm.)	0.6 (1.52 cm)

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TABLE I.—*Continued.*

Draft—main flue beyond boilers, in inches of water,	1.5–2 (3.81–5.08 cm.)	0.9 (2.29 cm.)	1.5 (3.81 cm.)	1.65 (4.19 cm.)	1.25 (3.18 cm.)		1.75″ (4.45 cm.)
Dimensions of main flue	15′ x 20′ (4.57 x 6.10 m.)	115 sq. ft. 10′ x 12′ (3.04 x 3.66 m.)	20′ x 48′ (6.1 x 14.63 m.)	19′–8″ x 22′–6″ (5.99 x 6.86 m.)	29′–3″ x 8′–6″ (8.91 x 2.59 m.)		18′ x 20′ (5.49 x 6.1 m.)
Dimensions of chimney	300′ x 30′ (91.44 x 9.14 m.)	187′ x 12′–6″ (57 x 3.81 m.)	506′ x 50′ (154.22x15.23m.)	300′ x 30′ (91.44 x 9.14 m.)	300′ x 15′ (91.44 x 4.57 m.)		350′ x 25′ (106.68 x 7.62m.)
Height of top of stack above grate	740′ (225.54 m.)	198′ (60.35 m.)	768′ (234.07 m.)	504′ (153.61 m.)	316′–9–1-2″ (96.55 m.)		412′ (125.57 m.)
Temperature of gas at bridge	2,800° F. (1,538° C.)	2,700°–2,800° F. (1,482°–1,538° C.)	2,650° F. (1,455° C.)	2,600°–2,800° F. (1,427–1,538° C.)	2,700°–3,000° F. (Est.) (1,482°–1,649° C.)		2,600° F. (1,427° C.)
Temperature of gas at throat	2,200° F. (1,205° C.)	2,300°–2,400° F. (1,260°–1,316° C.)	2,550° F. (1,397° C.)	2,000° F. (1,094° C.)	2,200° F. (approx.) (1,205° C.)		2,000° F. (1,093° C.)
Temperature of gas at far side of boilers	680° F. (360° C.)	500° F. (260° C.)	(No boilers.)	900° F. (482° C.)	800° F. (approx.) (427° C.)		775° F (412° C.)
Average boiler h-p. per furnace from waste heat	560 (568 C.V.)	971 *a* (984 C.V.)	None.	330 (335 C. V.)	486 (493 C.V.)	632 (641 C.V.)	390 (396 C. V.)
Per cent. fuel recovered as steam	32.81	56.62	None.	27.00	32.3	33.8	41.2
Per cent. fuel recovered from ashes	7.50	None.	5.00	9.50	5.05	.....	None.
Ratio of concentration	4.79	4.96	4.30	3.50	2.31	3.40	9.00
Tons charge per million B. t. u.	0.182	0.1574	0.115	0.147–0.167	0.122	0.171	0.165

a Economizers are installed at this plant beyond boilers.

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TABLE II.—*Analysis of Materials Used and Produced.*

		Copper.	Silver.	Gold.	SiO_2.	Fe and Mn.	CaO.	S.	Al_2O_3.
		Per Cent.	Oz. Per Ton.	Oz. Per Ton.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Charge	Anaconda	9.31	6.43	0.028	28.63	26.46	3.50	7.34	5.22
	Cananea	6.43	2.22	0.14	26.96	31.34	2.58	9.59	5.54
	Great Falls	9.47	3.77	0.0236	22.50	26.91	4.20	11.40	6.40
	Garfield	12.00	3.50	0.16	30.00	22.50	11.50	10.50	4.50
	Steptoe (McGill)..								
	Coal-fired	13.60	0.401	0.078	26.80	25.90	5.80	9.50	5.30
	Oil-fired	13.10	0.345	0.078	27.60	24.10	5.40	9.30	6.90
	Tooele	2.90	3.80	0.11	30.50	35.00	3.80	6.50	.....
Slag	Anaconda	0.39	0.19	0.0006	39.68	32.83	4.62	0.91	7.08
	Cananea	0.35	0.11	.....	37.44	33.58	3.83	1.08	7.96
	Great Falls	0.35	0.12	Trace.	42.50	23.64	11.60	0.48	9.40
	Garfield	0.45[a]	0.08	Trace.	46.50	20.58	17.50	1.00	5.50
								App'x	
	Steptoe (McGill)..								
	Coal-fired	0.45[a]	0.06	0.005	40.20	28.90	10.50	0.40	7.60
	Oil-fired	0.50[a]	0.02	0.007	42.00	25.90	10.90	0.20	8.00
	Tooele	0.43[a]	0.45	0.008	40.50	36.98	4.00	.....	.....
Matte	Anaconda	41.68	28.40	0.130	0.27 (Insl.)	26.54	.....	25.76	.....
	Cananea	31.91	11.34	0.07	0.54	37.17	.....	26.87	.....
	Great Falls	31.96	12.00	0.065	No data	35.54	.....	26.70	.....
	Garfield	42.00	10.00	0.50	.....	27.00	.....	26.00	.....
	Steptoe (McGill)..								
	Coal-fired	41.76	1.25	0.25	0.70	29.00	.....	25.10	.....
	Oil-fired	42.99	0.91	0.22	0.50	29.10	.....	24.80	.....
	Tooele	23.00	27.50	0.85	0.50	43.00	.....	26.50	.....

[a] Converter-slag is poured into reverberatories at these plants, thus raising copper-assay, as only 75 per cent. of copper in converter-slag is recovered.

		Moisture.	V.C.M.	Fixed Carbon.	Ash.	B.t.u. (Dry.)	B.t.u. (Wet.)
		Per Cent.	Per Cent.	Per Cent.	Per Cent.		
Fuel	Anaconda	6.13	36.28	45.42	12.17	12,390—	11,710—
	Cananea					No data.	18,600—
	Great Falls	7.70	23.70	46.00	28.60	No data.	9,382—
	Garfield	8.00	39.40	46.40	6.20	No data.	12,000—
	Steptoe McGill: Coal-fired	2.00	39.10	51.50	7.40	13,000—	.....
	Oil-fired					18,220—	.....
	Tooele	6.00	38.00	45.00	11.00	.....	11,500

	CO_2.	CO.	CH_4.	H.	N.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Gas—Great Falls	10.50	15.80	2.20	13.70	57.80

The following are data regarding the Kyshtim furnaces:

Surface of air checker, 869 sq. m.; useful area, 688 sq. m.; sectional area of air checker, 49 sq. m.; surface of gas checker, 698 sq. m.; useful area, 559 sq. m.; sectional area of gas checker, 46 sq. meters.

Average gas-analysis: CO_2, 7.47; CO, 26.22; H, 8.3; CH_4, 5.86; C_2H_4, 0.79 per cent.

Analysis of products of combustion in stack: CO_2, 12.2; O, 6.3; N, 71.5 per cent.; SO_2, ?

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Pressure at reversing-valve, 3 mm. of water. Suction at stack, 25 millimeters.

Temperature of gas entering regenerator, 68° C. Temperature entering stack, 385°C.

Average temperature in furnace, 1,600° C. Maximum temperature at ports, 2,000° C. Temperature entering checker, 1,300° C.

The charge was mainly flue-dust, but there was some green ore, some siliceous oxidixed copper-ore, some converter-cleanings, etc., mixed with it. I cannot give the composition of the charge, but 90 per cent. of it was flue-dust and green fines.

From January to September, 1910, the furnace was in operation 166 days, with results as follows:

Amount smelted, tons	10,690
Wood consumed in producers, cords	3,417
Matte produced, tons	3,326
Contents of matte, tons of copper	283
Assay, Cu, per cent	8.5
Contents of charge, copper, tons	330.8
Average assay of charge, Cu, per cent	3.1

The fuel is chiefly interesting for the extremely low fuel-cost, which has been verified by a first-class engineer on the ground; wood at Kyshtim costs $1.05 per cord delivered, so that the average fuel-cost over the nine months' period was 33.6 cents per ton of 2,000 lb. of charge. It comes about from the fact that the local Russians in charge of the furnace have had a lot of experience with gas-producers in iron-works, and made a gas of very high caloric value.

An analysis of a sample of Kyshtim flue-dust taken from one month's production gives: SiO_2, 8.8; Fe, 50.9; Cu, 3.3; S, 9.6 per cent.

A sample from a 6,000-ton pile of flue-dust gives: SiO_2, 8.9; Fe, 48.9; Cu, 3.3; S, 10 per cent.

	Per Cent.
A sample of one week's production gave Cu	3.38
A sample of large pile (weight not given) gave Cu	3.2
A sample of large pile, made in 1907, gave Cu	1.8

An average sample of raw ore used with flue-dust gave: SiO_2, 2.1; Fe, 38.5; CaO, 0.5; S, 47.0; Cu, 3.2 per cent.

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The data in regard to the Kyshtim plant have been recently communicated by John H. Allen.

At Cananea, Texas oil has been used; and at Humboldt, fuel-oil from California. At the plant of the Steptoe Co., at McGill, Nev., fuel-oil from California was introduced during 1911, with gratifying results.

A great many different varieties of burners were tried, but a simple home-made burner (arranged as shown in Fig. 2), using air for atomizing the oil, has given the best results.

A record performance at the Steptoe plant, at McGill, is communicated by Superintendent Sörensen: on Dec. 11, 1911, No. 1 furnace smelted 666 tons of total charge on an oil-consumption of five-eighths of a barrel of oil per ton of charge.

Charge—Analysis.

Total charge per furnace day, tons	666
Oil fired per furnace day, bbl.	421
Coal-equivalent of oil fired, tons	124.0
Total charge per bbl. of oil, tons	1.58
Oil, bbl. per ton of total charge	0.63
Equivalent gross coal as per cent. of total charge	18.60

Charge—Components as Per Cent. of Total Charge.

	Per Cent.
Calcines	60.1
Seconds	16.8
Converter hot slag	9.0
Fettling	3.9
Lime-rock	9.6
Flue-dust	0.6

Slag—Assay and Analysis.

	Per Cent.
Cu	0.40
SiO_2	44.0
Fe	34.3
CaO	8.6
Al_2O_3	7.4
Oxygen-ratio	2.72
Grade of matte, Cu	40.4

Draft of Water—In Inches.

Bridge	0.35
Throat	0.83
Stack	1.25

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FIG. 2.—OIL-FIRED REVERBERATORY AT STEPTOE PLANT.

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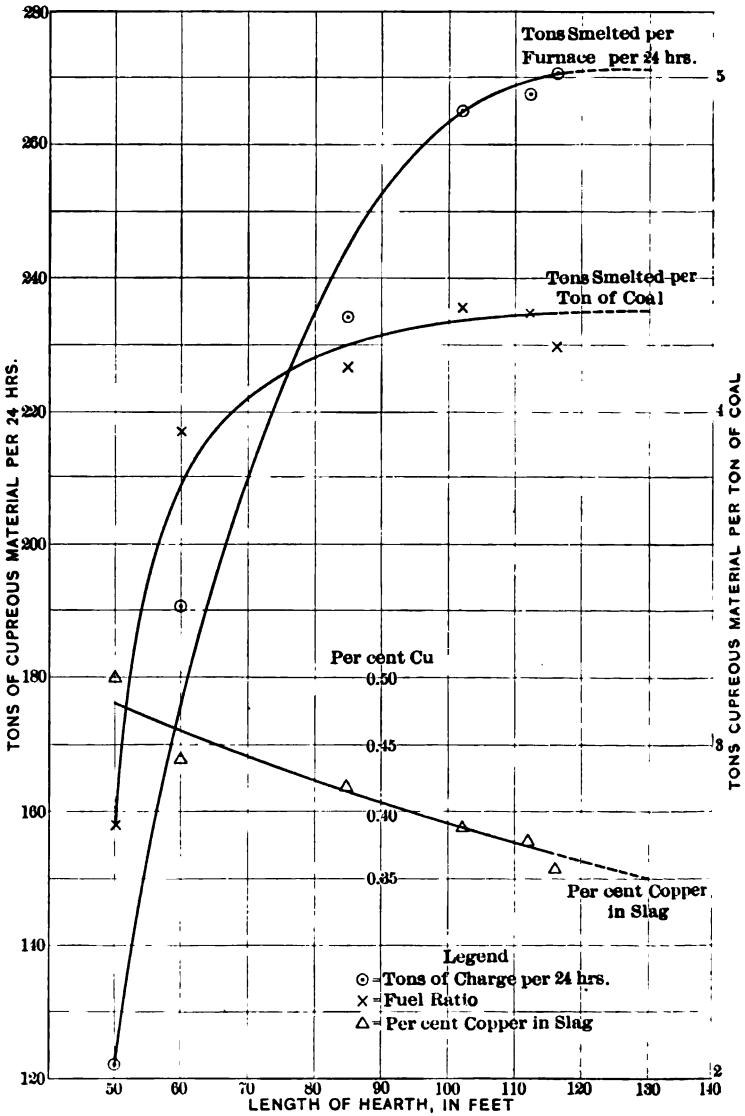

FIG. 3.—DIAGRAM SHOWING TONNAGE- AND ASSAY-CURVES, ANACONDA FURNACE.

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Temperature, Etc.

Temperature at verb	1,910° F.
Infusibility-factor of calcines	1.5
Burners, large Steptoe, high-pressure	7

At the Washoe plant, in Anaconda, coal is used for fuel, and a record run of one furnace was the smelting of 402.5 tons of charge in 24 hr., with a fuel-ratio of 1 ton of fuel for 6.45 tons of charge.

At Great Falls, Mont., the fuel is producer-gas, but the producers are located too far away from the furnaces to get the best results; new furnaces are contemplated for this plant, in

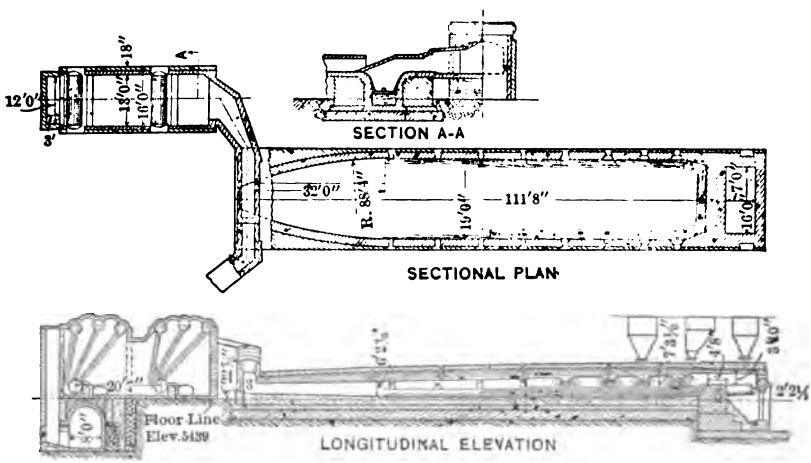

FIG. 4.—PLAN AND ELEVATION OF ANACONDA REVERBERATORY FURNACE AND BOILERS. ANACONDA COPPER MINING CO.

which the producers will be placed in close proximity to the furnaces and a much greater efficiency will be obtained.

The Canadian Copper Co. is experimenting with pulverized coal as fuel, but the furnaces have not been in operation long enough to give data of any great value. Supt. David H. Browne writes that he is greatly pleased with the performance of the furnaces up to date.

The objections to the use of pulverized coal in reverberatory practice have been the settling of ash and unburnt fuel on the top of the charge and the clogging of the flues with ashes. The Canadian Copper Co. has a rather peculiar condition, in that the charge is basic, and that the addition of silica, in the

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form of coal-ash, is rather welcome than otherwise, and to avoid the other objection the coal is pulverized extremely fine, and the surplus ash goes out with the gaseous products of combustion.

At nearly all modern plants waste-heat boilers have been installed, and the diagrams, Figs. 4 to 18, show the important variations of these installations.

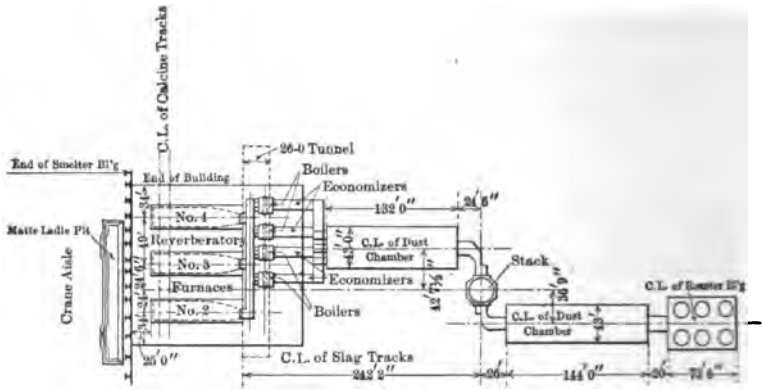

FIG. 5.—GENERAL ARRANGEMENT OF ROASTER AND REVERBERATORY PLANT. COPPER QUEEN CONSOLIDATED MINING CO.

Silica brick is used almost exclusively in the construction of the lining and roof of the modern reverberatory furnaces for copper-smelting, and it is now possible to obtain silica brick, of excellent quality, all over the United States at reasonable cost.

I desire here to thank the following gentlemen for valuable data supplied: Dr. E. D. Peters, Dr. H. O. Hofman, Dr. L. D. Ricketts, C. B. Lakenan, S. Severin Sörensen, George B. Lee, William H. Howard, A. E. Wheeler, William Wraith, H. N. Thomson, John H. Allen, David H. Browne.

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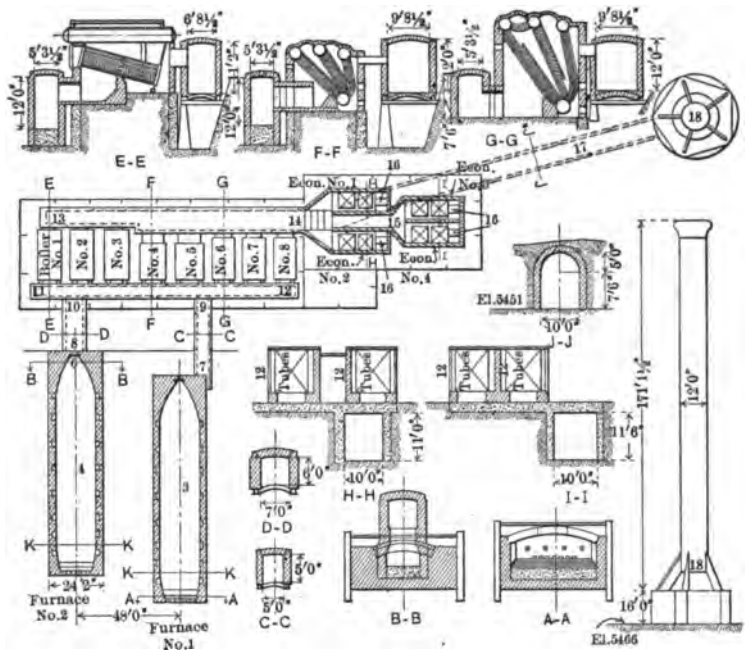

General Data.

	Area, Sq. Ft.
Two reverberatory furnaces, 100 by 19 ft. Hearth, *K. K.*	
Seven burner-holes, 6 by 10 in.	3
Four peep-holes, 10-in. diameter	2
Throat of furnace No. 1, 5 ft. wide, 2 ft. 8 in. high	14
Throat of furnace No. 2, 7 ft. wide, 1 ft. 9 in. high	14
Furnaces Nos. 1 and 2, throats	28
Cross-over flue No. 1, 5 ft. wide, 5 ft. high, 50 ft. long	28
Cross-over flue No. 2, 7 ft wide, 6 ft. high, 27 ft. long	44
Flue to boiler at 11	60
Flue to boiler at 12	40
Flue from boilers at 13	76
Flue from boilers at 14	120
Boiler No. 1, area entering, 34 sq. ft.; leaving	18
Boiler No. 2, area entering, 34 sq. ft.; leaving	18
Boiler No. 3, area entering, 34 sq. ft.; leaving	18
Boiler No. 4, area entering, 38 sq. ft.; leaving	20
Boiler No. 5, area entering, 38 sq. ft.; leaving	20
Boiler No. 6, area entering, 30 sq. ft.; leaving	19
Boiler No. 7, area entering, 30 sq. ft.; leaving	19
Boiler No. 8, area entering, 30 sq. ft.; leaving	19

	Area, Sq. Ft.
Boilers. Total area entering, 288 sq. ft.; leaving	151
Boilers Nos. 1, 2, 3, Aultmann & Taylor, 250 h-p., each.	
Boilers Nos. 4 and 5, Stirling, 250 h-p., each	
Boilers Nos. 6, 7, 8, Stirling, 300 h-p., each.	
Economizer No. 1, area entering, 83 sq. ft.; leaving	45
Economizer No. 2, area entering, 83 sq. ft.; leaving	45
Economizer No. 3, area entering, 83 sq. ft.; leaving	45
Economizer No. 4; area entering, 83 sq. ft.; leaving	45
Underground flue	115
Stack, 12 ft. 6 in., 171 ft. 1¼ in. high, base.	123
Smallest area for passing gases, 14 sq. ft; largest area	123
California crude oil used, S. G., 0.966.	
Weight per barrel of 42 gal., 339 lb.	
Heat-content per lb., 18,700 B.t.u.	
Economizers, Green Economizer Co., 288 tubes.	
Seven oil-burners per furnace. 0.43 lb. of steam required to atomize 1 lb. of oil.	
Total travel of gases in furnace No. 1.... 690 ft.	
Total travel of gases in furnace No. 2....730 ft.	
	Draft.
Temperature of gases at 9 and 10, 2,300°–2,400° F.	0.25
Temperature of gases at 14, 500° F.	0.60
Temperature of gases at 18, 350° F.	0.90

FIG. 6.—SMALL SCALE ARRANGEMENT OF REVERBERATORIES, BOILER CONNECTION, ECONOMIZERS AND STACK. CANANEA CONSOLIDATED COPPER CO.

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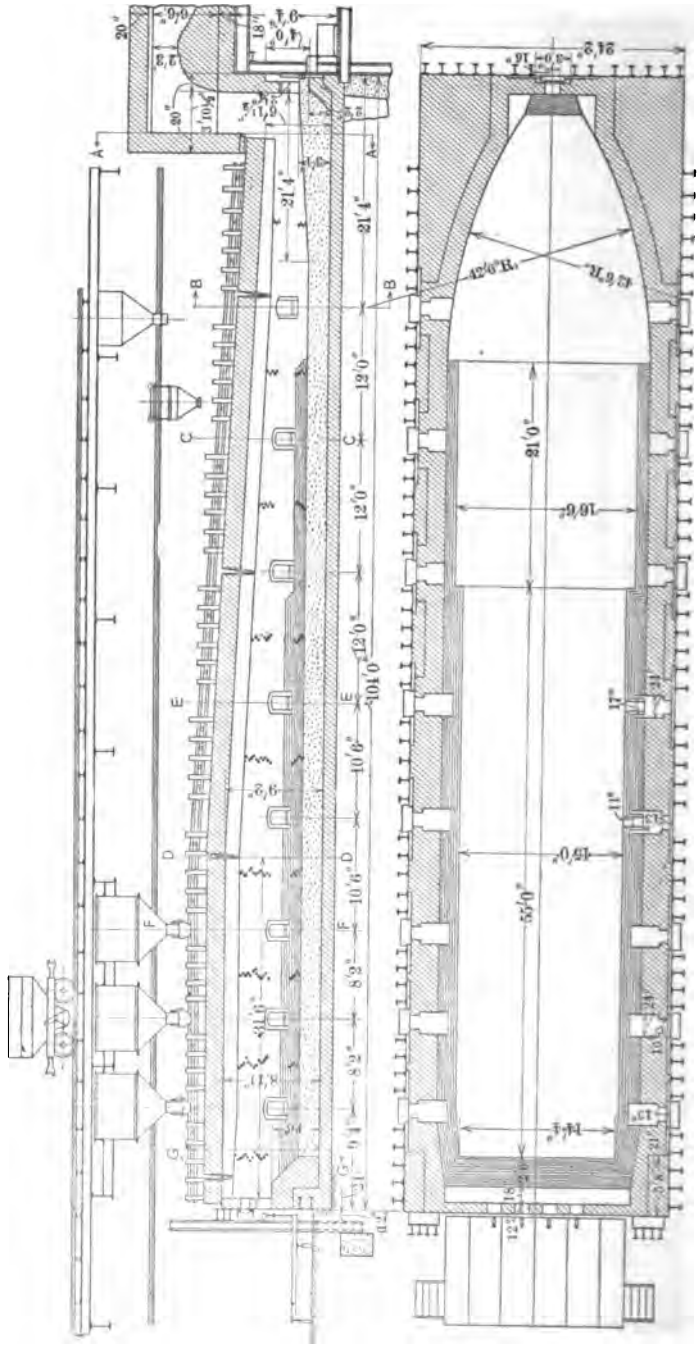

FIG. 7.—OIL-FIRED FURNACE. PLAN AND ELEVATION. CANANEA CONSOLIDATED COPPER CO.

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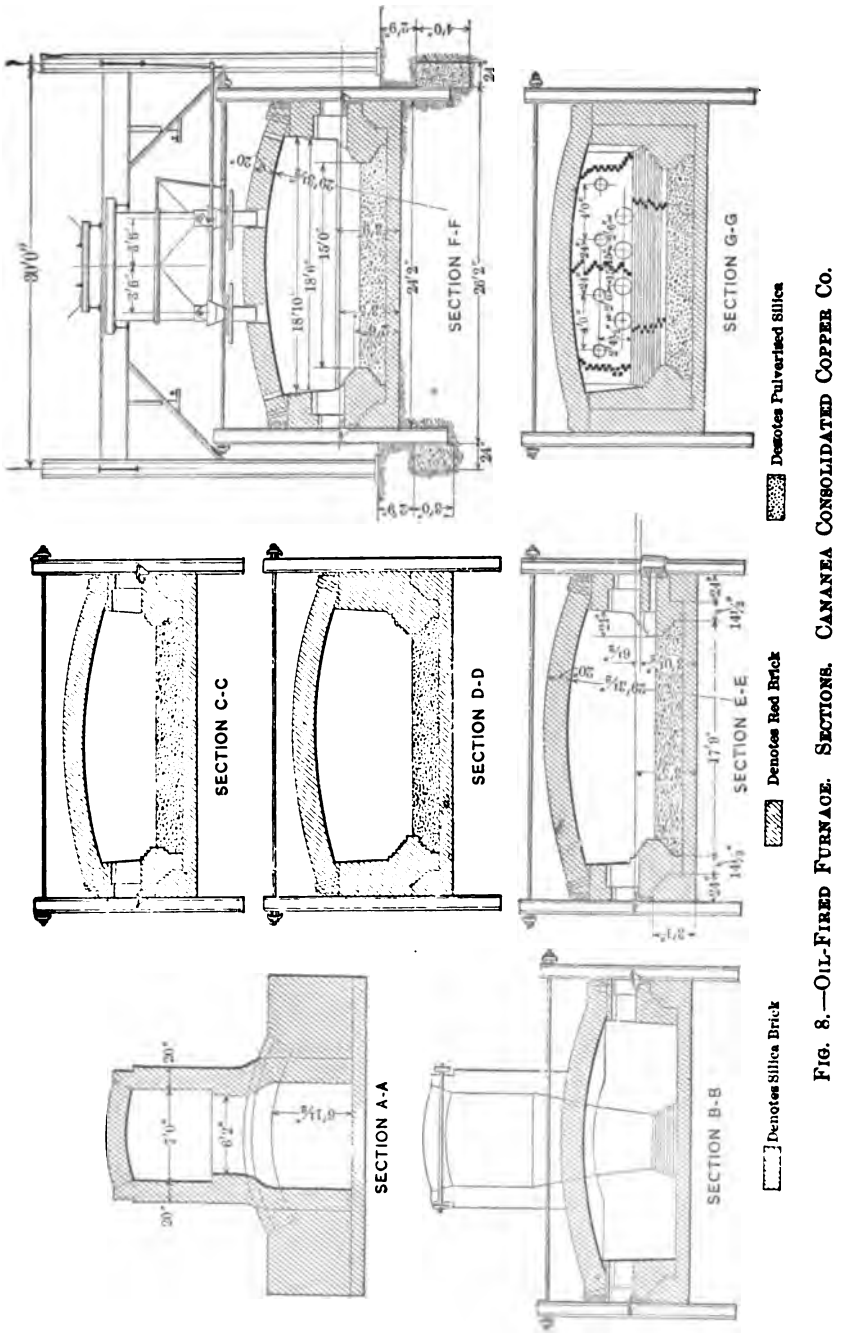

FIG. 8.—OIL-FIRED FURNACE. SECTIONS. CANANEA CONSOLIDATED COPPER CO.

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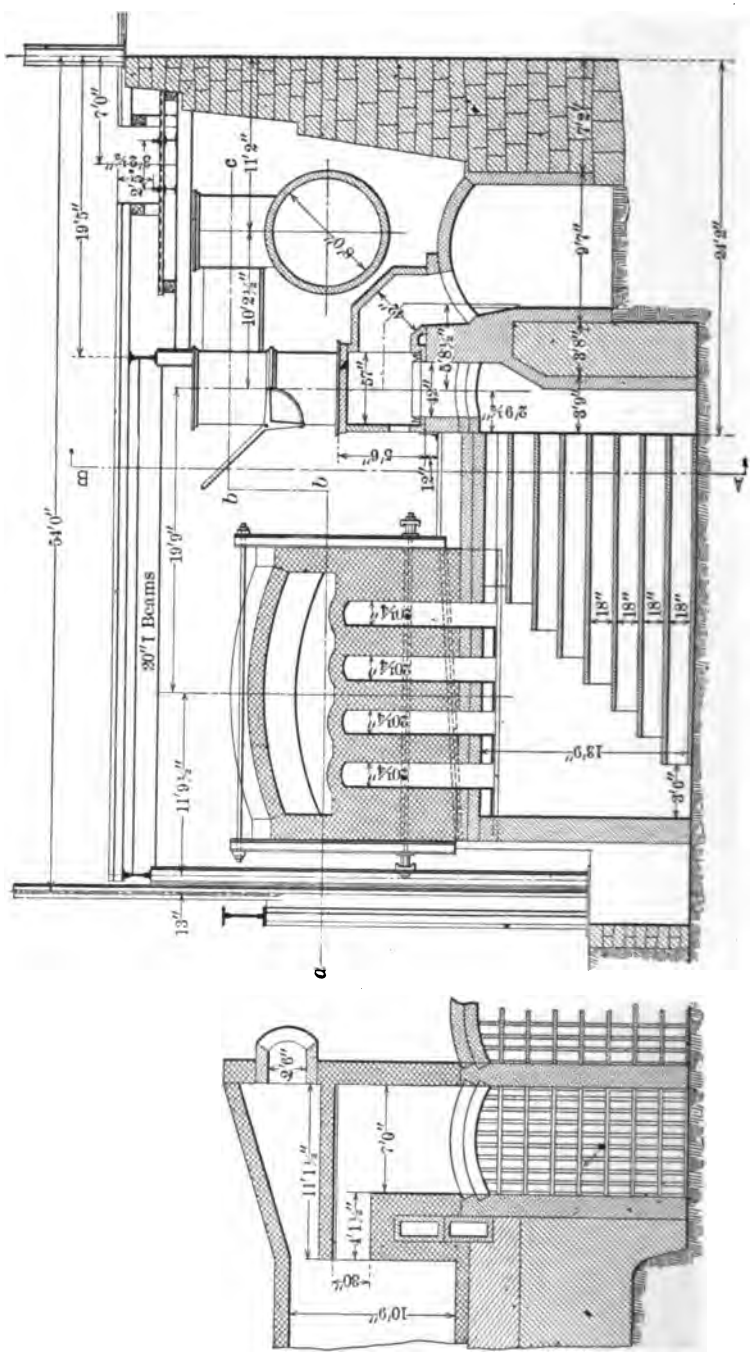

FIG. 9.—CHECKER-WORK AND SECTION THROUGH FLUES. BOSTON & MONTANA REDUCTION DEPARTMENT.

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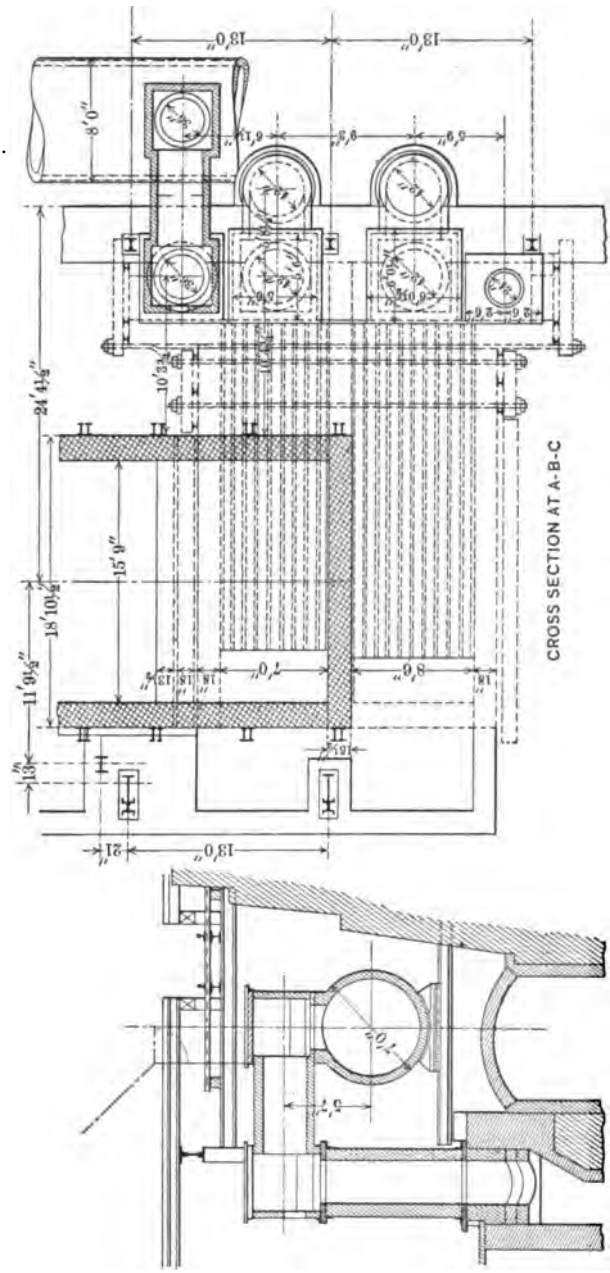

FIG. 10.—SECTIONS OF FIG. 9.

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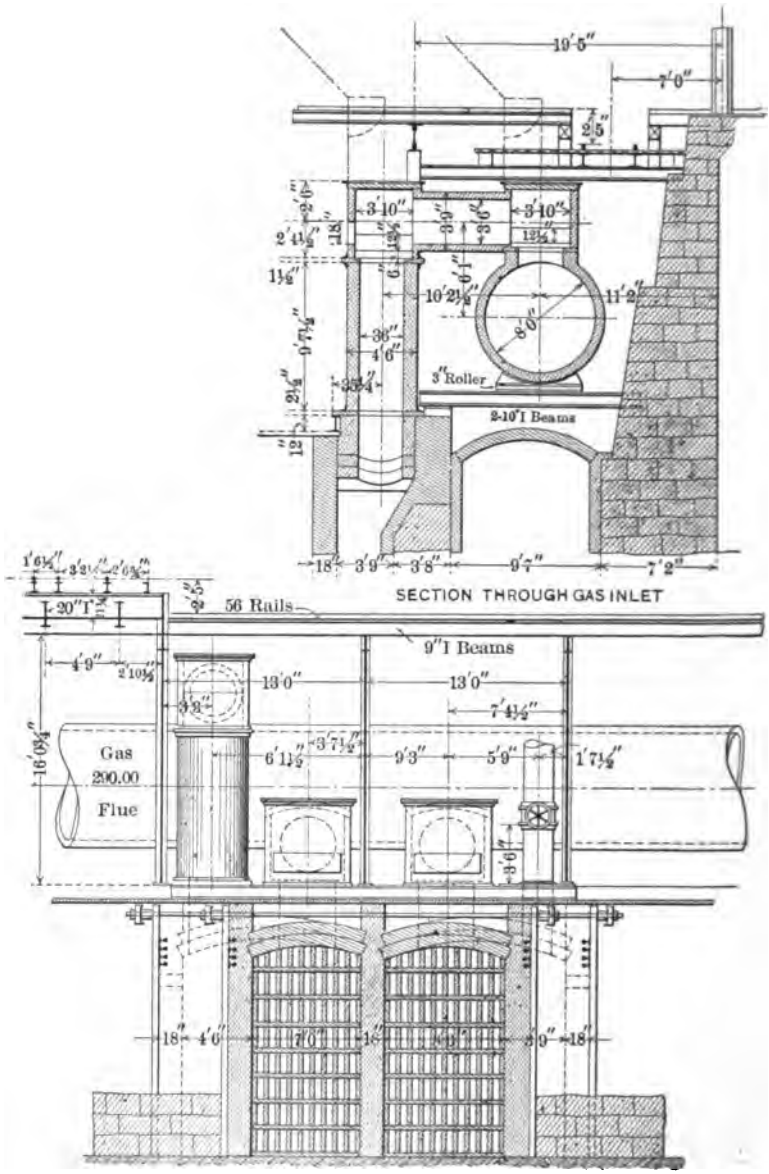

SECTION AT A-B

FIG. 11.—SECTIONS OF FIG. 9.

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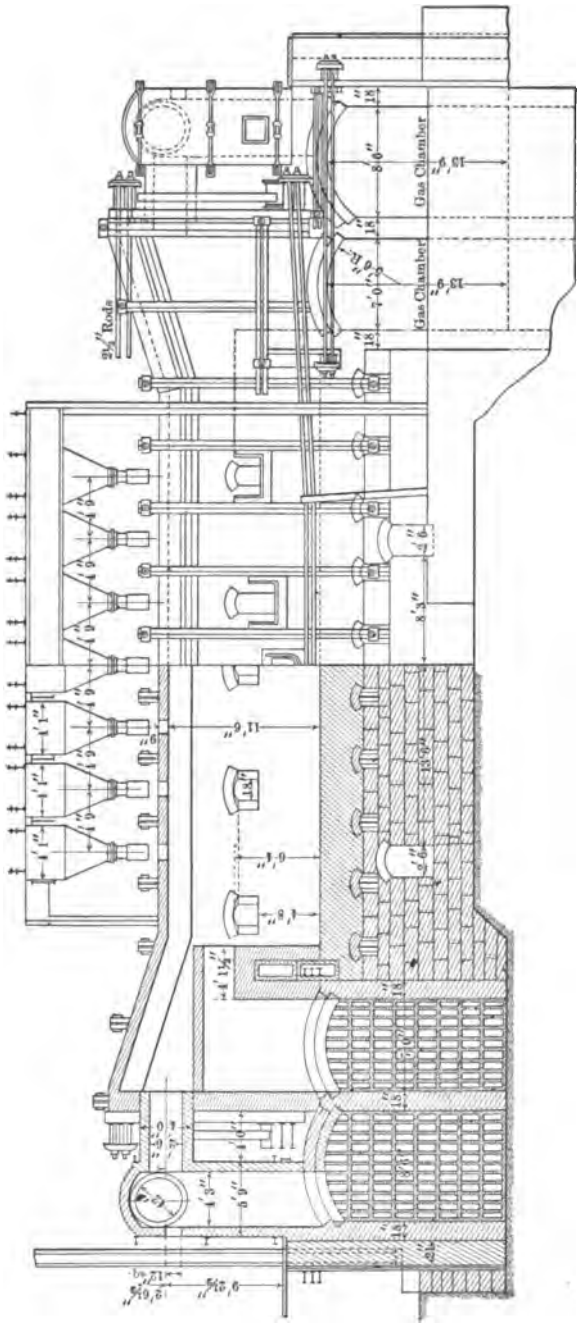

FIG. 12.—GAS-FIRED FURNACE. ELEVATION. BOSTON & MONTANA REDUCTION DEPARTMENT.

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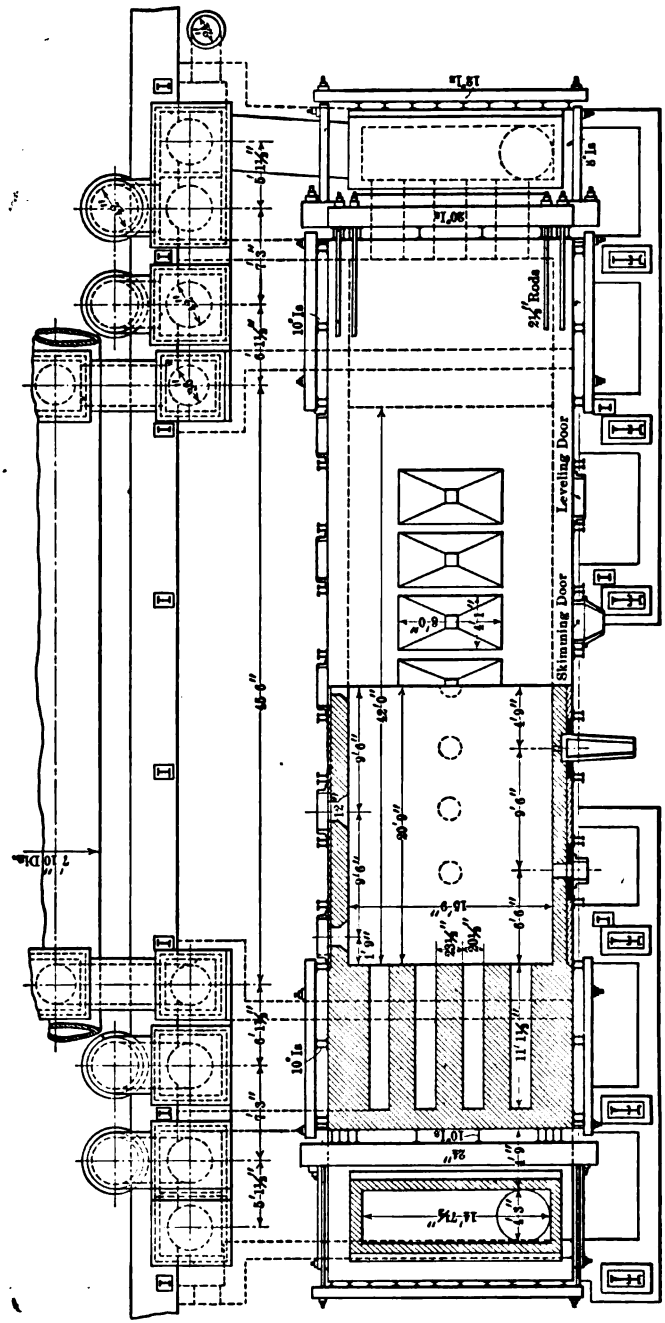

FIG. 18.—GAS-FIRED FURNACE. PLAN. BOSTON & MONTANA REDUCTION DEPARTMENT.

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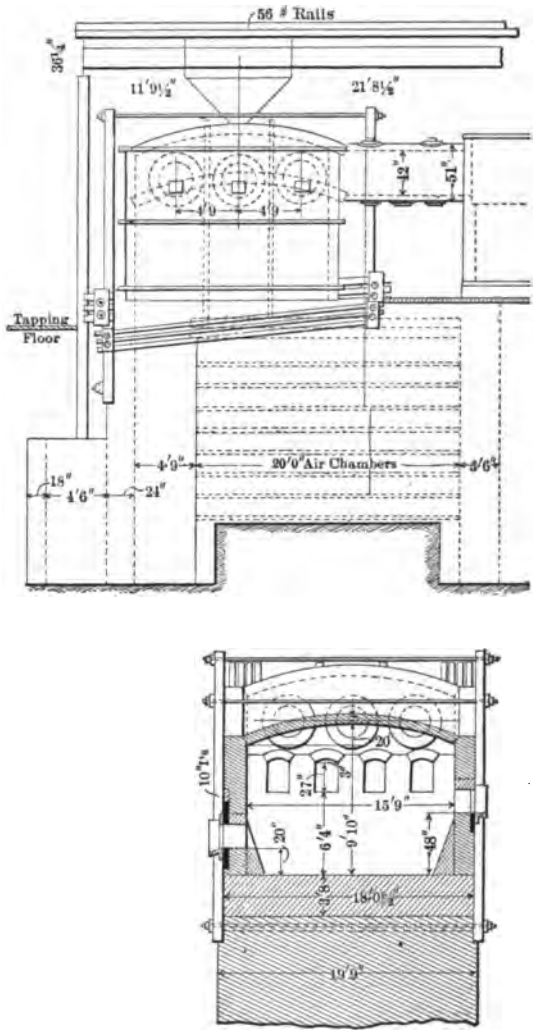

FIG. 14.—GAS-FIRED FURNACE. SECTIONS. BOSTON & MONTANA REDUCTION DEPARTMENT.

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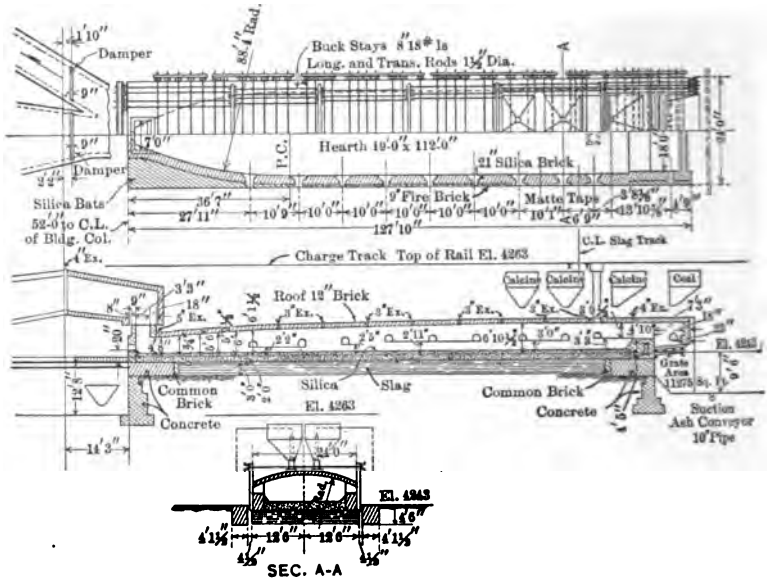

FIG. 15.—PLAN AND ELEVATION OF FURNACE. AMERICAN SMELTERS SECURITIES CO.

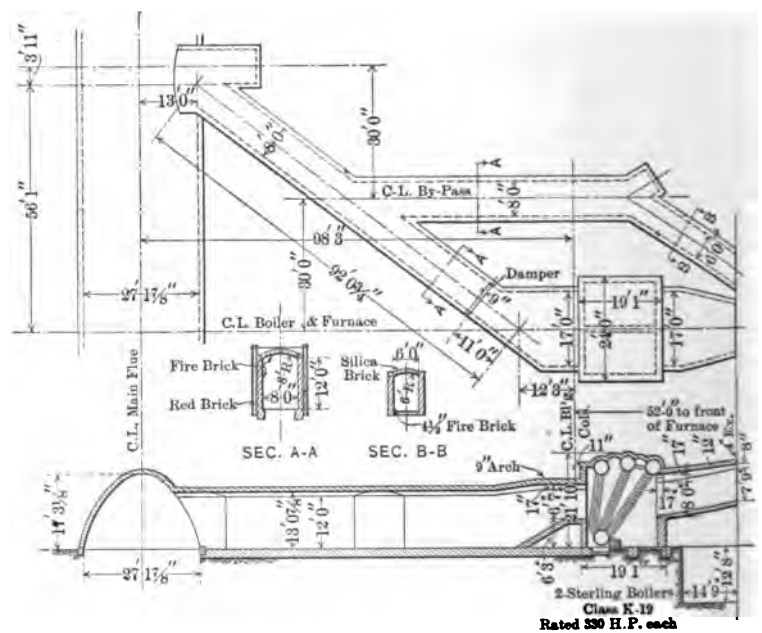

FIG. 16.—FLUE AND BOILER ARRANGEMENT. AMERICAN SMELTERS SECURITIES CO.

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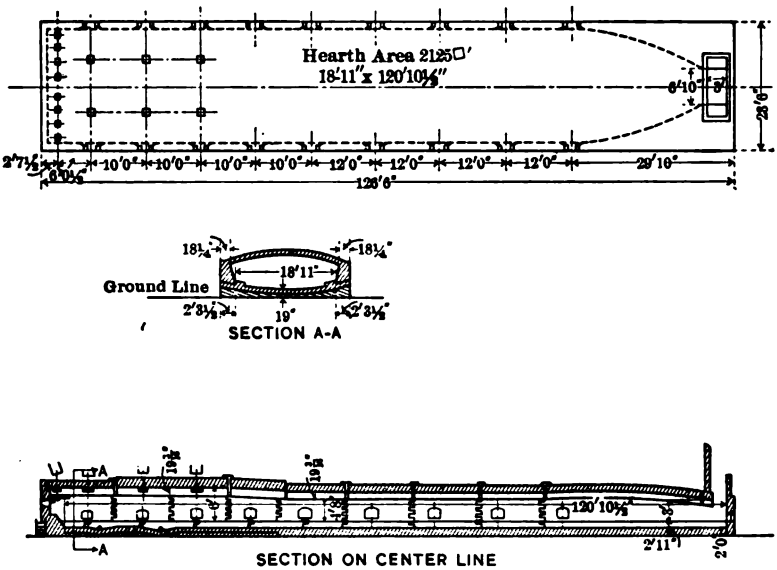

FIG. 17.—PLAN AND ELEVATION OF NO. 1 FURNACE. STEPTOE, MCGILL, NEVADA.

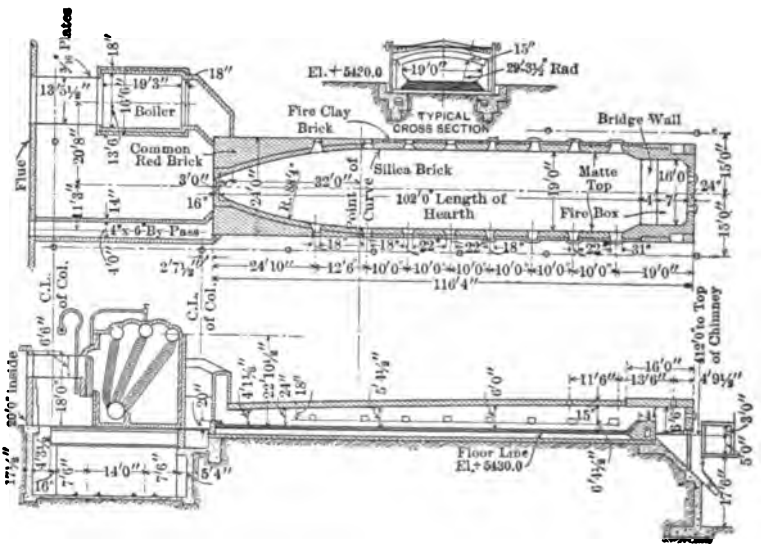

FIG. 18.—PLAN AND ELEVATION OF FURNACE AND BOILER. INTERNATIONAL SMELTING & REFINING CO.

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Chemistry of the Reduction Processes in Use at Anaconda, Mont.*

BY FREDERICK LAIST, ANACONDA, MONT.

(Cleveland Meeting, October, 1912.)

THE ores received at the Washoe Smelter come almost entirely from the mines in Butte and contain the following minerals:

Chalcocite, Cu_2S; covellite, CuS; chalcopyrite, $CuFeS_2$ (trace); bornite, Cu_3FeS_3; enargite, Cu_3AsS_4; malachite, $Cu_2CO_4 + H_2O$ (trace); pyrite, FeS_2; sphalerite, ZnS; rhodochrosite, $MnCO_3$ (less than 1 per cent.); rhodonite, $MnSiO_3$ (less than 1 per cent.).

The country-rock of Butte is of the granite family and is a quartz-monzonite containing a little less silica than typical granite. It is made up of the following rock-minerals in approximately the proportions indicated:

Minerals.	Composition.	Per Cent.
Quartz	SiO_2	20
Andesine	$\frac{1}{2}$ $NaAlSi_3O_8$; $\frac{1}{2}$ $CaAl_2Si_2O_8$	42
Orthoclase	$KAlSi_3O_8$	23
Hornblende	$\frac{1}{2}$ $Ca(MgFe)_3Si_4O_{12}$; $\frac{1}{2}$ $(MgFe)_2(AlFe)_4Si_2O_{12}$	5
Biotite	$(HK)_2(MgFe)_2Al_2Si_3O_{12}$	9
Titanite	$CaTiSiO_5$	0.1
Apatite	$(CaF)Ca_4(PO_4)_3$	0.05
Magnetite	$(FeO)(Fe_2O_3)$	0.15
Pyrite	FeS_2	0.02
		99.32

The main country-rock (quartz-monzonite) is cut by dikes of aplite, a granite having the following composition: Quartz, 30; andesine, 20; orthoclase, 45; ferro-magnesian minerals, 3; pyrite, 0.1; total, 98.1 per cent.

Closely associated with the copper-deposits are dikes and bodies of "Modoc" porphyry, technically known as quartz-

* Presented also at the joint meeting of the Institute and Section IIIa, Metallurgy and Mining, of the Eighth International Congress of Applied Chemistry, New York, N. Y., September, 1912, and here published by mutual agreement.

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porphyry and having the following composition: Quartz, 24; andesine, 40; orthoclase, 25; ferro-magnesian minerals, 6; pyrite, 0.4; total, 95.4 per cent.

The following is an analysis of a typical altered quartz-monzonite occurring in immediate contact with the copper-bearing veins:[1]

	Per Cent.
SiO_2	62 09
Al_2O_3	15.49
Fe_2O_3 and FeO	8.52
MgO	0.42
CaO	0.20
Na_2O	0.37
K_2O	4.34
H_2O at 110° C.	1.20
H_2O above 110° C.	3.01
TiO_2	0.51
P_2O_5	0.13
S	5.47
Cu	0.25
Total	102.00
Less oxygen	2.05
	99.95

How the various rock-components may be combined is immaterial so far as the purposes of the present paper are concerned. For instance, in the furnace, silica will combine with iron and lime to form a slag whether it occurs as quartz, andesine, orthoclase, or any other mineral.

It does, however, make a difference in the coal- and coke-consumption of the furnaces, and on the grade of matte produced by the blast-furnaces, whether the silica charged is free or combined. The former is less and the latter higher when the silica is mostly present as quartz. This is, of course, to be expected and follows from thermo-chemical considerations.

It would be beyond the scope of the present paper to trace out all of the reactions entered into by all of the components of the ores, and, furthermore, such an attempt would be largely guess-work.

Following is a list of minerals and compounds the reactions of which are responsible for by far the larger part (say 95 per

[1] *Bulletin No. 228, U. S. Geological Survey* (1904).

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cent.) of the activity in the various furnaces, and the writer intends to confine himself to discussing the reactions going on among them.

Chalcopyrite	Cu_2S	Sphalerite	ZnS
Covellite	CuS	Hematite	Fe_2O_3
Bornite	Cu_3FeS_3	Iron oxide	FeO
Enargite	Cu_3AsS_4	Limestone	$CaCO_3$
Pyrite	FeS_2	Silica	SiO_2
		Alumina	Al_2O_3

The departments of the reduction-works which are of interest to us are as follows: 1, concentrator; 2, roaster; 3, reverberatory; 4, blast-furnace; 5, converter.

The chemistry of each of these departments will be taken up in the order named, excepting the concentrator, which has no chemistry. The function of the concentrator, it may be said in passing, is purely to wash out enough of the siliceous gangue of the ore (with a minimum loss of copper, silver, and gold) to leave a concentrate containing just the right proportions of silica and iron for the subsequent operation of smelting. The concentrate is, therefore, merely an enriched ore, and differs chemically from the ore in no respect except that it contains a larger percentage of copper, iron, and sulphur and a smaller percentage of silica, alumina, and other gangue-constituents.

The products of the concentrator which will be referred to in the following pages are coarse and fine concentrates and slimes. The slimes consist of ore which has been too finely ground for successful concentration, together with the clay-like decomposition-products of the rocks. This slime is caught in settling-ponds, allowed to dry partly, and smelted in the blast-furnaces after being briquetted together with fine concentrates and fine ore.

Roasting-Furnace Reactions.

All concentrates which are finer than 5 mm. are known technically as fine concentrates and go into the MacDougall roasting-furnaces. This material makes up over 90 per cent. of the charge, the remainder being fine ore and fine limestone. The latter undergoes no change in the roasters, the temperature being insufficient to cause dissociation into CaO and CO_2.

The concentrates contain about 33 per cent. of sulphur, and

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about 75 per cent. of this must be roasted out in order to yield a calcine suitable for reverberatory smelting. The calcine contains about 8 per cent. of sulphur. The concentrates (as well as fine ore) may be regarded as essentially a mixture of gangue and FeS_2, Cu_2S, Cu_3FeS_3, and Cu_3AsS_4.

On the first hearth the concentrates are dried. On the second hearth the roast commences with the breaking-down of some of the FeS_2 as follows:

$$\text{(a)}\ 7\ FeS_2 = Fe_7S + 6\ S.$$
$$\text{(b)}\ 6\ S + 12\ O = 6\ SO_2.$$

At this point and this temperature the copper sulphides are not materially affected. On the following hearths the temperature is gradually increasing to a strong red heat and the various constituents are roasting according to the following reactions:

$$FeS + 3\ O = FeO + SO_2.$$

When too much air is admitted, the calcine becomes red, owing to oxidation of FeO, thus:

$$2\ FeO + O = Fe_2O_3$$

The copper sulphides undergo the following reactions:

$$Cu_2S + 4\ O = 2\ CuO + SO_2$$
$$CuS + 3\ O = CuO + SO_2$$
$$Cu_3FeS_3 + 10\ O = 3\ CuO + FeO + 3\ SO_2$$

The enargite breaks up as follows:

$$2\ Cu_3AsS_4 + 25\ O = 6\ CuO + As_2O_3 + 8\ SO_2$$

All of the arsenic, however, is not driven out, as a portion unites with copper to form an arsenite, or arsenate, which does not decompose until it gets into the reverberatory furnace, where a portion of its arsenic goes into the slag and another portion into the matte.

In the reactions given so far the formation of sulphates has been ignored. This takes place as follows:

$$FeS + 4\ O = FeSO_4$$

It is uncertain whether the oxygen combines directly with FeS to form $FeSO_4$ or whether the sulphate is formed indirectly as follows:

$$SO_2 + O\ (\text{in presence of } Fe_2O_3) = SO_3$$
$$SO_3 + FeO = FeSO_4$$

Fe_2O_3 is known to be a good catalytic agent; therefore, a part at least of the iron sulphate is formed in this manner.

Copper sulphate occurs in the calcine and is partly formed

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the same as iron sulphate and partly on the hotter hearths by the decomposition of iron sulphate.

$$FeSO_4 = FeO + SO_3$$
$$CuO + SO_3 = CuSO_4$$

Iron sulphate is considerably less stable than copper sulphate. Therefore, if a mixture of $FeSO_4$ and CuO are heated together to a temperature intermediate between the decomposing-temperatures of $FeSO_4$ and $CuSO_4$, the iron sulphate will break up and the liberated SO_3 will immediately unite with CuO.

Reverberatory-Furnace Reactions.

A reverberatory furnace is essentially a melting-furnace, the atmosphere of which is approximately neutral. Hence, the material fed into it must have been oxidized to the desired degree in the roasters: *i. e.*, sufficient iron sulphide must have been oxidized to iron oxide to form a suitable slag with the silica present.

The heat required for melting is generated by the combustion within the furnace of coal, wood, oil, or gas. The active constituents of the first three named fuels are carbon and hydrogen combined as hydrocarbons. Fuel-gas is a mixture of CO_2, CO, H, CH_4 (and other hydrocarbons), and N. The CO_2 and N are inert. CH_4 is formed by distillation from the coal in the gas-producer. In the furnace it burns thus:

$$CH_4 + 4\ O = CO_2 + 2\ H_2O$$

The CO and H are formed by the action of oxygen (air) and steam respectively on the incandescent carbon of the coal.

(a) $C + 2\ O = CO_2$
(b) $CO_2 + C\ (\text{red hot}) = 2\ CO$
(c) $H_2O + C\ (\text{red hot}) = 2\ H + CO$

Heat is generated in the furnace by the re-combination of H and CO with oxygen.

(d) $2\ H + O = H_2O$
(e) $CO + O = CO_2$

Reactions (a), (d) and (e) are exothermic. Reactions (b) and (c) are endothermic. Reaction (c) is often used to reduce the temperature in fire-box or producer when the coal clinkers badly or to increase the length of flame in the furnace where the coal used is deficient in volatile hydrocarbons.

The charge to the furnaces may be considered to be a mixture

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of gangue, the principal constituent of which is SiO_2 with some Al_2O_3, and FeS, FeO, Fe_2O_3, Cu_2S, CuO, $CaCO_3$, with small quantities of $CuSO_4$ and $FeSO_4$. At the temperature prevailing in the furnace the limestone is first decomposed thus·

$$CaCO_3 = CaO + CO_2$$

The iron and calcium oxides unite with the silica to form a complex silicate which constitutes about 86 per cent. of the slag. The simplest silicate reactions are :

$$FeO + SiO_2 = FeSiO_3$$
$$CaO + SiO_2 = CaSiO_3$$

The slag which is being made at Anaconda corresponds very nearly to the formula $\begin{pmatrix} 8\ FeO \\ CaO \end{pmatrix}$. $9\ SiO_2$ and is a bi-silicate.

The particular type of slag to be made is a matter of expediency and of the action of the slag on the furnace-lining. It is necessary, periodically, to fettle reveberatory furnaces: *i.e.*, to throw silica against the fire-bridge and side-walls, along the slag-line, for a distance of about 15 ft. back. This is necessary, owing to the tendency of the slag to cut: *i.e.*, dissolve silica from the walls of the furnace where the temperature is highest. The particular type of silicate which will form is largely a matter of temperature. Hence, in order to reduce cutting to a minimum, we must proportion our charge to make a slag which is practically saturated with silica at the particular temperature in our furnace.

Of the remaining 14 per cent. of the slag, the principal constituent is Al_2O_3 (about 7 per cent.). Just how the alumina behaves, whether as an acid or as a base, has been a most fruitful subject of discussion among metallurgists. I believe the general opinion is that it is capable of filling both rôles, according to whether the surrounding medium is acid or basic in its character. Thus, in the presence of an excess of base, the alumina is likely to combine with it, forming aluminates, while in the presence of an excess of silica it will play the part of base and we will have silicate of aluminum in the slag. Both series of compounds, however, seem to be difficultly fusible and are viscous and gummy at the temperature ordinarily attained in smelting-furnaces, and these characteristics are imparted to the slag. In the presence of much alumina it is not possible to run successfully as siliceous a slag as in its absence. Aluminates of

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calcium and of magnesium, of the formulæ $Al_2O_3.CaO$ and $Al_2O_3.MgO$, have been made by heating mixtures of Al_2O_3 with CaO and MgO, respectively, in an electric furnace. Fire-clay is a well-known silicate of aluminum.

The remaining 6 or 7 per cent. of the slag consists of MgO, MnO, Cu, S, alkalies, carbon, etc.

So much for the slag-formation. The reactions which yield matte and influence its grade are dependent, first of all, upon the power of sulphur combined with iron to reduce copper from combination as oxide or silicate, thus:

$$4\ FeS + 6\ CuO = 4\ FeO + 3\ Cu_2S + SO_2$$

The simple reaction:

$$FeS + CuO = CuS + FeO$$

is impossible, owing to the instability of CuS at high temperature, this compound breaking down into $Cu_2S + S$. Copper silicate, chrysocolla for example, is reduced similarly with formation of iron silicate:

$$4\ FeS + 4\ CuSiO_3 = 4\ FeSiO_3 + 2\ Cu_2S + 2\ S$$

Thus sulphur acts as an excellent protective agent, preventing loss of copper in the slag.

It has been said under roasting that Fe_2O_3 forms to a certain extent, particularly when too much air is admitted during the operation of roasting. Much has been said as to the harmful effect of this, due to the difficult fusibility of ferric silicate slags. It is my opinion that this effect cannot be very serious in the presence of sulphides, since these would reduce ferric to ferrous oxides thus:

$$3\ Fe_2O_3 + FeS = 7\ FeO + SO_2$$

Practically all of the iron sulphide which is not decomposed by one of the last three reactions goes into the matte and it is for this reason that it is necessary to adjust the calcination so as to leave just enough sulphur to cause the formation of, say, a 40-per cent. Cu matte. Otherwise, the matte will be too low-grade, due to the presence of too much FeS, and, on the other hand, the slag be too siliceous, owing to the absence of FeO.

Small quantities of sulphates present in the calcines break up in the fierce heat of the reveberatory.

$$CuSO_4 = CuO + SO_2 + O$$
$$FeSO_4 = FeO + SO_2 + O$$

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Blast-Furnace Reactions.

The copper blast-furnace, smelting sulphidé ores, combines the functions of roasting- and melting-furnaces: *i.e.*, it oxidizes and melts simultaneously, incidentally utilizing the heat generated by the oxidation of sulphur and iron for melting.

The ore, concentrates, and briquettes charged to the blast-furnace we may consider a mixture of gangue, consisting principally of SiO_2 (with some Al_2O_3), and FeS_2, Cu_2S, and $CaCO_3$. (Other compounds of sulphur and copper behave in an analogous manner to Cu_2S, and oxides of copper or iron may be left out of consideration, owing to the small quantities of these which are received.) The converter-slag which constitutes about 12 per cent. of the charge is an iron silicate containing copper, probably also present as silicate. It has the formula $2\ FeO.SiO_2$. Coke is charged into the furnace to supply whatever heat cannot be obtained from oxidation of Fe and S.

In the upper part of the charge column a certain amount of roasting goes on and FeS_2 and $CaCO_3$ break up, with liberation of S and CO_2, respectively. The utilization of the oxygen blown through the tuyeres of a blast-furnace is very perfect and very little free oxygen is found immediately over the top of the charge. This is shown by the clouds of yellow sulphur which make their appearance whenever a fresh charge is dropped on a hot top. There is not enough O present to burn the S set free from FeS_2.

The oxidation (Bessemerizing) of the sulphides and coke, as well as the union between oxides and silica to form slag, takes place mainly in the tuyere-zone. The reactions are the same as those taking place in roasters and reverberatories, but proceed with much greater intensity and rapidity.

(a) $FeS + 3\ O = FeO + SO_2$.
(b) $C + 2\ O = CO_2$.

The Anaconda blast-furnace slag corresponds closely to the formula:

$$\left.\begin{matrix}(7\ FeO)\\(9\ CaO)\end{matrix}\right\} 12\ SiO_2$$

and is a sesqui-silicate.

From reactions (a) and (b), we can readily see that an excess of coke in the sulphide-smelting furnace must have an injurious effect. The amount of oxygen in a blast-furnace is limited and

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what there is will most readily combine with the carbon present before going to FeS.

Therefore, an excess of carbon (coke) reduces the amount of iron available for slag by preventing oxidation of FeS. Furthermore, it drives the latter into the matte and lowers its grade with respect to copper.

The converter-slag, as has been said, corresponds to the formula $2\,FeO.SiO_2$ and is a uni-silicate. At the temperature prevailing in the smelting-zone of the blast-furnace it is capable of combining with more silica to form a bi- or sesqui-silicate, thus:

$$(FeO)_2.SiO_2 + SiO_2 = 2\,FeO.SiO_2$$
$$2\,(FeO)_2.SiO_2 + SiO_2 = (FeO)_4.3\,SiO_2$$

Any silicate of copper which may be in the converter-slag or which may be in the blast-furnace charge is reduced by FeS according to the reaction given under reverberatory smelting.

Converter Reactions.

The charge to the converters consists of molten matte. Matte may be regarded as $Cu_2S.FeS$, although the matte received at the converters averages about 7 per cent. less in copper than is called for by this formula.

The linings of the present converters are magnesite brick, and the silica required to flux the iron oxide formed by the action of the air-blast on FeS must, therefore, be supplied by addition of ore. The reactions during the first half of the "blow" are as follows:

$$FeS + 3\,O = FeO + SO_2$$
$$2\,FeO + SiO_2 = 2\,FeO.SiO_2\ (slag)$$

The slag is a uni-silicate of approximately the formula given.

It is important that the slag be poured out of the converter as soon as all the iron (all but about 1.5 per cent.) is oxidized. Otherwise, the charge will foam out through the mouth. The exact reason for this phenomenon is not quite clear, but it is probably due to the formation of copper silicate, which thickens the slag and prevents free passage of the air. (It has been suggested that the foaming of the slag may be partly due to excessively rapid evolution of gases from the converter-charge at about the time when the iron is all oxidized. That the elimination of sulphur is very rapid at about this period of the blow

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has been proved by several series of samples taken from the converter at 10-min. intervals. An abnormally rapid evolution of gases in conjunction with a thickening of the slag would readily account for the foaming phenomenon.)

$$CuO + SiO_2 = CuSiO_3$$

The copper oxide, and therefore, silicate, cannot form in the presence of much FeS, and for this reason, no trouble is encountered from this reaction till the iron sulphide is nearly all gone.

The slag having been removed, oxidation of the copper sulphide commences:

$$Cu_2S + 4\ O = 2\ CuO + SO_2$$

The copper oxide immediately reacts with sulphide:

$$Cu_2S + 2\ CuO = 4\ Cu + SO_2$$

The progress of the various reactions is made manifest to the converter-men by changes in the flame color. This is green while the iron is oxidizing, blue just before removing the slag, and red while the Cu_2S is blowing to copper. The end-point of the charge can hardly be told from the color of the flame, however, and other indications are made use of, a discussion of which does not come within the scope of the present paper.

The above are the main and essential reactions in the converter. The matte always contains impurities, the chief ones at this plant being lead, zinc, and arsenic.

Experiments made here have shown that these are oxidized and eliminated as fume and in the slag throughout the slag-forming period of the blow, and this elimination is practically complete in the case of lead and zinc, and nearly so in the case of arsenic. Very little elimination of arsenic takes place during the second, or copper-forming, period.

In closing, a word may be said as to the reactions in the refining- and casting-furnace. The copper is poured into this furnace in the molten state as it comes from the converter and it contains about 98.5 per cent. of copper, together with the silver and gold for which the copper has served as a collector, and fractional quantities of sulphur and iron. Here the copper is blown with air, which oxidizes these and also some of the copper. The copper is now brought back to the proper "pitch"

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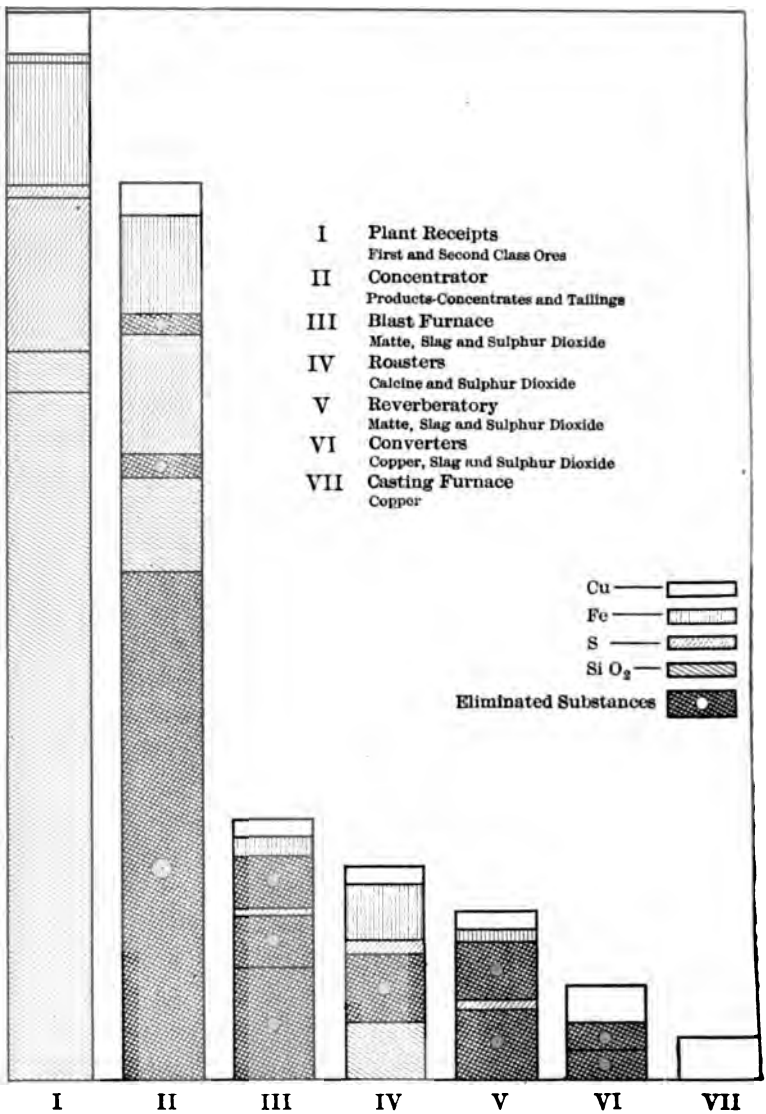

FIG. 1.—CHART SHOWING THE ELIMINATION OF THE IRON, SILICA, AND SULPHUR IN THE REDUCTION PROCESSES AT ANACONDA.

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by poling, which reduces the CuO. The impurities form a slag on the surface, which is collected by means of cinders and skimmed off. It may be of interest to note, in passing, that the arsenic-elimination in the Anaconda casting-furnaces is very slight, this being almost entirely completed in the converters.

I append herewith a chart, Fig. 1, which shows graphically the elimination of the valueless constituents of the ore in the various operations. The widths and lengths, respectively, of the blocks are proportional to copper tonnages and cupreous material tonnages treated in the various plants. The blocks are subdivided according to the quantities of copper, iron, sulphur, and silica entering each plant. The cross-hatched areas show what is eliminated in each plant.

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The Sulphatizing-Roasting of Copper-Ores and Concentrates.*

BY UTLEY WEDGE, ARDMORE, PA.

(Cleveland Meeting, October, 1912.)

In general, the art of securing copper from sulphide ores or concentrates may be said to consist of: (1) separation, in the molten state, of copper sulphide with some iron sulphide, from the great bulk of gangue-material, followed by (2) elimination of sulphur and iron from the molten matte, by oxidation and fluxing, leaving metallic copper.

The definition is made broad enough to cover the general practice, whether or not the ore or concentrate is given a preliminary roast for the removal of surplus sulphur.

The above process, which I will designate as the smelting process, makes necessary the presence in the ore, or the addition thereto, of such silica, lime, and iron as may be necessary for the formation of a light and fusible slag, together with sufficient sulphur and iron for the formation of copper-matte, and also the addition to the matte of sufficient silica for the oxidation and fluxing-out of the iron.

These items, together with the cost of fuel sufficient for melting the mass, constitute important items of cost in the copper-smelting process.

Numerous cases where fluxing-materials are scarce or dear, or where fuel is expensive, have caused special attention to be given recently to the wet process of copper-extraction, which may be designated as consisting of forming soluble copper salts which are then washed or leached out of the ore or concentrates by dissolving in water or acid. Metallic copper may then be secured from the solution of copper salts by electrolysis; by cementation; or copper can be brought out of the solution as a hydroxide or other insoluble precipitate and reduced to metallic copper by subsequent melting in a reducing atmosphere.

* Presented also at the Joint Meeting of the Institute and Section IIIa, Metallurgy and Mining, of the Eighth International Congress of Applied Chemistry, New York, N. Y., September, 1912, and here published by mutual agreement.

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The present paper presents facts relating to only one step in the wet process of copper-extraction and deals only with the formation of copper sulphate by roasting copper-ores or concentrates.

One per cent. of copper-content requires, theoretically, 0.5 per cent. of sulphur to form copper sulphate. In practice, it is necessary to have present an excess of sulphur over this theorectical amount. If sufficient sulphur is not present in the material to be treated, it is then necessary to add sulphur in some form, either as sulphur, SO_2, SO_3, or in the form of a sulphide or sulphate or sulphuric acid. Iron sulphides are very desirable in ores, when it is required to sulphatize the copper values, and zinc, lead, lime, and magnesia, if present in forms that will consume sulphur, are undesirable.

The principal reactions which enter into the sulphatizing of copper sulphides are as follows:

CuS decomposes in roasting to $Cu_2S + S$, and the free atom of S burns to SO_2 at temperatures above 350° C. (662° F.). Cu_2S at furnace-temperatures below 500° C. (932° F.) reacts principally as follows;

$$2\ Cu_2S + 5\ O_2 = 2\ CuO + 2\ CuSO$$

At higher temperatures, the principal reaction is

$$Cu_2S + 2O_2 = 2\ CuO + S_2$$

Different copper sulphide minerals behave differently at furnace-temperatures, but the above formulas show the tendency common to all copper sulphides to form some quantity of copper oxide at all furnace-temperatures.

In roasting to secure the highest per cent. of water-soluble copper, the constant formation of the copper oxide is the chief obstacle to overcome.

The presence of iron sulphide with copper sulphide in the roasting process assists in sulphatizing the copper oxide thus formed. At furnace-temperatures below 600°C. (1,112°F.) there is some basic iron sulphate formed, which, at temperatures above 530°C. (986°F.), roasts to Fe_2O_3 and SO_3, and the SO_3 gas may combine with the CuO to form $CuSO_4$, which is practically stable at temperatures below 650°C. (1,202°F.), or, the CuO may react directly with the $Fe_2(SO_4)_3$, according to the formula,

$$3\ CuO + Fe_2(SO_4)_3 = Fe_2O_3 + 3\ CuSO_4$$

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Even the portion of iron sulphide which roasts to Fe_2O_3 is of some assistance in sulphatizing the CuO, as at a temperature between 500° C. and 750° C. the iron oxide acts as an energetic catalyzing agent for the oxidizing of the SO_2 to SO_3, which in turn tends to combine with the CuO to form $CuSO_4$.

Also, SO_2 generated in the furnace, either from the copper or iron sulphides, tends to sulphatize the copper oxide.

These various principles are generally familiar from enunciation by Aubell, Kothny, Ottokar Hofmann and others, and attention is called to them here to make clear the significance of the following tests made by the Wedge Mechanical Furnace Co., of Philadelphia. The tests were made without the addition of any material to the ore treated:

Test No. 621, *February*, 1910.

Analysis of Ore.

	Per Cent.
Cu	1.07
Fe	49.25
S	32.60
SiO	6.80
MgO	2.18
Al_2O_3	1.59

The copper present was chalcopyrite; iron present was pyrite and pyrrhotite.

Ore was roasted in a Wedge multiple-hearth furnace, with the following results:

	Per Cent.
Water-soluble copper	0.35 = 27.8 per cent. extraction.
Acid-soluble copper	0.80 = 63.4 per cent. extraction.
Insoluble copper	0.11
Total	1.26 = 91.2 per cent extraction.

The calcine from the above roast contained 4.7 per cent. of sulphur and was then treated in a Wedge multiple-hearth mechanical muffle-furnace. It was then leached, with the following result:

	Per Cent.
Water-soluble copper	0.75 = 60.3 per cent. extraction.
Acid-soluble copper	0.48 = 38.1 per cent. extraction.
Insoluble copper	0.02

Total copper secured in solution 98.4 per cent.

In the above test, the copper designated as acid-soluble copper was copper oxide, soluble in very dilute sulphuric acid.

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It will be observed that, in spite of the presence of liberal quantities of iron sulphide, there was considerable formation of copper oxide in the furnace, but this occurred chiefly in the preliminary roast, where, by reason of the considerable excess sulphur, the temperature became excessive. Analysis of the gas leaving the furnace in initial roasting showed formation of sufficient SO_3 for the production of sulphuric acid to recover all of the copper oxide, or, if electrolysis were used in subsequent recovery of the copper from solution, more sulphuric acid than would be required to sulphatize the copper oxide.

Test No. 620, *March*, 1910.

Analysis of Ore.

	Per Cent.
Cu	3.52
Fe	9.76
S	9.54
Insol	66.5

Copper present as chalcopyrite; gangue consisted of schist.

This ore was given a single treatment in a Wedge multiple-hearth mechanical muffle-furnace 9 ft. 9 in. in diameter with five hearths; four hearths muffle-fired with coal.

The analyses made currently during the test showed:

Water-soluble copper, from 60 to 70 per cent. of total copper.
Acid-soluble copper, from 20 to 22 per cent. of total copper.

Total, from 80 to 92 per cent. of total copper.

In this test the SO_2 and SO_3 gases generated in the upper hearths of the furnace were drawn downward over the ore on the lower hearth of the furnace.[1]

If electrolysis were used, the copper sulphate solution would yield, in this case, a great excess of sulphuric acid for the recovery of the copper oxide.

Test No. A–100, *July*, 1911.

Analysis of Ore.

	Per Cent.
Cu	4.38
Fe	9.61
S	16.37
CaO	1.10
MgO	0.72
Insol	61.21

[1] U. S. Patent No. 966,277, Aug. 2, 1910.

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Copper present as chalcopyrite; the gangue was diabase.

This ore was given a single treatment in a Wedge multiple-hearth muffle-furnace[2] having a diameter of 9 ft. 9 in. with five hearths, being the same furnace as was used in the previous test.

Actual leaching of this ore showed a recovery by

Water extraction	73.7 per cent. of copper in ore.
Weak sulphuric acid	15.7 per cent. of copper in ore.
Making a total of	89.4 per cent. of the total copper in ore.

It is believed that this ore, sulphatized in a larger furnace of the Wedge type, would yield still better results.

Tests so far given, were made with ores containing copper as chalcopyrite.

The following tests were made with an ore in which the copper was present chiefly as bornite, with some copper silicate and carbonate.

Sample No. 25, *April*, 1911.

Analysis of Ore.

	Per Cent.
Cu	3.40
Fe	11.05
S	13.02
SiO_2	63.90

This ore was ground to 20-mesh and was given a single treatment in a small furnace test and showed:

Water-soluble copper	80.80 per cent. of total copper-content.
Acid-soluble copper	11.8 per cent. of total copper-content.
Total recovery	92.6 per cent. of total copper-content.

The above results have since been duplicated with this ore in a Wedge mechanical muffle-furnace 20 ft. in diameter with five hearths, and sufficient sulphuric acid is recovered from the SO_3 in gases leaving the furnace to recover the oxide of copper formed and make the extraction as above shown in the subsequent leaching-operation.

Sample No. 7, *December*, 1909.

Analysis of Ore.

	Per Cent.
Cu	1.85
Ni	2.75
Fe	47.78
S	30.85
SiO_2	15.30

[2] U. S. Patent No. 654,335, July 24, 1900.

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It was desired to extract the copper, leaving the nickel in the leached ore, together with the iron.

The analysis of the calcine showed:

Water-soluble.....	Copper, 69.2 per cent.	Nickel.....	13.6 per cent.
Acid-soluble	Copper, 12.8 per cent.	Nickel.....	4.4 per cent.
Total soluble.....	Copper, 82.0 per cent.	Nickel.....	18.0 per cent.

The leached ore contained

Copper.....	0.29 per cent.
Nickel.....	2.80 per cent.

It has been shown in tests made by the Wedge Mechanical Furnace Co. that copper-ores containing little or no sulphur or iron can be treated by sulphatizing-roasting by the addition of iron sulphide in the form of iron pyrites.

Test No. 604, *August*, 1909.

Analysis of Ore.

	Per Cent.
Cu.....	3.14
Fe.....	2.25
S.....	none
Insoluble.....	85.5

This ore contained copper as carbonate and the gangue was porphyry.

This ore was mixed with 5 per cent. of iron pyrites containing 47.5 per cent. of sulphur.

The test was made in a 9 ft. 9 in. Wedge mechanical muffle-furnace with five hearths, and the analysis of the calcine showed 92 per cent. of the copper to be soluble in water. The copper solution was very pure, containing only from 0.1 to 0.2 per cent. of Fe_2O_3 to 7 per cent. of copper sulphate. Copper precipitated from the solution with clean scrap-iron tested 96 per cent. pure cement copper.

Test No. 634, *November*, 1910.

Analysis of Ore.

	Per Cent.
Cu.....	1.86
Fe.....	2.79
S.....	0.80
Al_2O_3.....	2.58
Insoluble.....	88.3
Combined water.....	3.4

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The sulphur contained in this ore seemed to be partly in cinders from the locomotive, which accumulated in the open car during transit from the West. Copper was present largely as copper silicate. Six per cent. of pyrites containing 47 per cent. of sulphur was added to this ore and this mixture was given a single treatment in a Wedge mechanical furnace, with five hearths and a preheating hearth, muffle-fired with coal; diameter of furnace 9 ft. 9 in. Tests were made of the material as it progressed through the furnace from hearth No. 1 (top hearth) and showed as follows:

Leaving Hearth No. 1 79.3 per cent. of copper water-soluble.
Leaving Hearth No. 2 81.4 per cent. of copper water-soluble.
Leaving Hearth No. 3 93.4 per cent. of copper water-soluble.
Leaving Hearth No. 4 95.2 per cent. of copper water-soluble.
Leaving Hearth No. 5 95.2 per cent. of copper water-soluble.

There was no heat applied to the fifth hearth and the sulphatizing was completed in the fourth hearth. This calcine leached readily with cold water, using 1 ton of water per ton of ore. Solution secured was used on successive batches and brought up to 15 per cent. of $CuSO_4$ containing only 0.1 per cent. of Fe. The tailings from this leaching-test contained Cu 0.08 per cent.

Up to this time, very few records are available from furnaces of large size. The furnace with which most of these records were made was a furnace 9 ft. 9 in. in diameter and 12 ft. high, with five hearths and a preheating hearth. This furnace, a generation ago, would have been considered a large furnace, even with a capacity of only one carload of ore per week, but the commercial problems of to-day call for much larger appliances. Several large mechanical muffle-furnaces are now being installed by large copper companies, and records from furnaces with an output of from 20 to 40 tons daily, per furnace, will soon be available.

The above is a record indicating in advance the possibilities of a process which may soon; in special cases, displace the smelting process of copper-production.

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Notes on the Metallography of Alloys.

BY WILLIAM CAMPBELL, PH.D., SC.D.,* NEW YORK, N. Y.

(Cleveland Meeting, October, 1912.)

[SECRETARY'S NOTE.—To avoid repetitions of foot-notes, references to authorities are made in the paper by means of figures, referring to a numbered list in the appendix.—J. S.]

I. INTRODUCTION.

IN the olden days the making of alloys was an art, and the secrets of the craft were jealously guarded. To-day it has become a science, though the old ideas in regard to the secrets and formulæ are dying hard.

Modern progress may be attributed to scientific research and testing of materials, and the substitution of chemical analytical control for "cook-book" methods. We know to-day a great deal about the constitution of alloys, the reasons of the change in structure and physical properties by heat and mechanical treatment; and this new knowledge is due for the most part to the physical chemist and metallographist, who have shown us that alloys are simply solutions, and as such obey definite laws. The application of the theories of solutions and of the phase rule tells us what ought to happen when our alloys are in a state of equilibrium. The working-out of the temperature-composition diagram (thermal diagram) for numerous alloys has thrown a light on the structure and constitution which has dispelled most of the darkness in which we labored before the advent of metallography.

The work began with the determination of the freezing-point curves, and here the research of Le Chatelier,[1] Gautier,[2] Roberts-Austen,[3] and the other pioneers broke the ground. Next the physical chemists, Tammann,[4] Rooseboom,[5] Bancroft,[6] and their colleagues, showed us the complete thermal diagram and its meaning. Then the researches of Heycock and Neville[7] on the Cu-Sn series and of Shepherd[8] on the

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Cu-Zn demonstrated the application to the most complex of alloys.

A vast amount of work has been published on the constitution of alloys, as will be seen from a glance at the pages of the *Zeitschrift für anorganische Chemie*, *Zeitschrift für physikalische Chemie*, *Metallurgie*, and *Revue de Métallurgie*. The diagrams have been collected notably by Portevin [9] in *Revue de Métallurgie*, Bornemann [10] in *Metallurgie*, and lastly in Guertler's *Text Book of Metallography*, now nearing completion.

The ternary diagram in the simplest case was worked out by Charpy [11] for the Pb-Sn-Bi series. More complicated cases have been given by Geer [12] and by Sahmen and Vegesack.[13]

The importance of bearing-metals in engineering makes no excuse necessary for a discussion of their constitution, and the following notes on the three groups of white metals (1) Pb-Sb-Sn, (2) Sn-Sb-Cu, (3) Zn base, and a comparison with the structure of the bronzes, may throw some light on the subject.

II. The Lead-Tin-Antimony Series.

The ternary alloys of Pb-Sn-Sb find considerable application in the arts as bearing-metal, type-metal, white metal for small castings, etc.

In regard to the composition of alloys used in practice, the list given in Table I. embraces the most important. A few alloys containing a small amount of copper are also given for comparison. The list shows that Sb never exceeds 25 per cent. Pb varies from 93 to 5 per cent. and Sn from 1 to 75 per cent. There is no uniform variation, however.

The physical properties of this series of alloys are not very well known, at least they have not been published in detail, and while the anti-frictional qualities within certain limits of use do not show the wide differences one would expect from the variation in composition, there is probably a fairly marked difference in crushing-strength as we pass from the lead-rich to the tin-rich side, judging from the structure. Charpy, in a study of white alloys called anti-friction,[14] examined some alloys of the Pb-Sn-Sb group in regard to crushing-strength and microstructure.

Before taking up the ternary alloys a few notes on the binary alloys will be given.

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TABLE I.—*Composition of Pb-Sn-Sb Alloys.*

No.	Name. Metal.	Pb.	Sn.	Sb.		Authority.
		P.C.	P.C	P.C.	P. C.	
1.	Electrotype-metal.,	93	3		.....	Private notes.
2.	Bearing-metal	82	1		.....	Private notes.
3.	Linotype-metal	85	3		.....	Private notes.
4.	Bearing	83.3	8.3	8.3	.....	Private notes.
5.	Stereotype-metal	82	6	12	.....	Private notes.
6.	Bearing	82	2	16	.....	Private notes.
7.	Stereotype	82	3.2	14.8	.....	Roberts-Austen.
8.	Bearing	80	10	10	.....	Private notes.
9.	Bearing—Compagnie de l'Est	80	12	8	.....	Charpy. Guillet.
10.	Bearing	80	5	15	.....	Guillemin. Guillet.
11.	Bearing—like "Glyco," etc	80.5	4.5	14.5	As, 0.5	Notes.
12.	Bearing—like "Magnolia"	78	6	16	.....	Notes.
13.	Bearing—Magnolia and Tandem	77.7	5.9	16.8	.....	Law. Thurston.
14.	Type metal	77.5	6.5	16	.....	Notes.
15.	Bearing—Anti-friction	77	10	12.5	Cu, 0.5	
16.	Bearing—like "Coleco"	77	8	14	Cu, 1.0	
17.	Bearing	76	7	17	.....	Notes.
18.	Metallic packing—Compagnie d'Orleans	76	14	10	.....	Guillet.
19.	Bearing—American R. R.	73.5	8	18.5	.....	Notes.
20.	Piston Packing—Compagnie du Nord	73	12	15	.....	Guillet.
21.	Bearing—French R. R.	70	20	10	.....	Charpy.
22.	Stereotype—Mackenzie metal	70	13	17	.....	Thurston.
23.	Bearing—Paris-Lyon-Mediterranée R. R.	70	10	20	.....	Guillet.
24.	Type	70	10	18	Cu, 2	Roberts-Austen.
25.	Bearing—American R. R.	68	21	11	.....	Notes.
26.	Bearing—Graphite-metal	68	15	17	.....	Dudley. Guillet.
27.	Stereotype	68	17	18	.....	Thurston.
28.	Type	63.2	12	24	Cu, 0.8	Notes.
29.	Bearing	62.5	26.2	10	Cu, 1.3	Notes.
30.	Bearing	62	27	10	.....	Notes.
31.	Type	60.5	14.5	24.2	Cu, 0.75	Notes.
32.	Type	60	35	5	.....	Notes.
33.	Bearing	60	20	20	.....	Brannt.
34.	Type—common	60	10	30	.....	Law.
35.	Solder	60	39	1	.....	Notes.
36.	Type	55.5	40	4.5	.....	Notes.
37.	Type—best	50	25	25	.....	Law.
38.	Bearing	48	40	10	Cu, 2	Notes.
39.	Bearing—American R. R., "No. 2"	46	36.5	16.5	Cu, 1	Notes.
40.	Hoyle's Metal	42	46	12	.....	Hiorns. Guillet.
41.	Bearing. Chemin de fer de l'Est Francais	42	42	16	.....	Guillet.
42.	Bearing	40	45	15	.....	Ledebur. Guillet.
43.	Bearing—German	40	42	26	Cu, 2	
44.	Bearing—Italian R. R.	37	38	25	.....	Thurston. Guillet.
45.	Stereotype	35	60	5	.....	Berthier. Thurston.
46.	White metal	33	53	10.6	Cu, 2.4	Zn 1. Roberts-Austen.
47.	Bearing	10	75	15	.....	Notes.
48.	Bearing	10	75	12	Cu, 3	Notes.
49.	For small castings	5	75	20	.....	Notes.

1. *Lead-Antimony.*

Sb freezes at 631° C., Pb at 327° C. The addition of Pb to Sb, or of Sb to Pb, lowers the freezing-point. The freezing-point diagram showing the relation of percentage-composition to temperature of freezing was worked out by Roland-Gosselin,[15] who found that the eutectic or alloy with the lowest freezing-point contained Sb, 13 per cent., and froze at 228° C.

Stead,[16] in a paper on the microchemical examination of Pb-Sb alloys, showed that the eutectic contained Sb, 12.8 per cent. and froze at 247° C. Gontermann[17] gave 13 per cent. and 245° C.

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The freezing-point or thermal diagram is shown in Fig. 1. Temperatures are represented in the vertical scale and percentage-composition in the horizontal. *A*, or 631° C., shows the freezing-point of Sb, which is lowered along the curve *Ar* as the proportion of lead in the alloy increases. Similarly, *B*, or 327 C., is the freezing-point of Pb, which is depressed along the curve *Br* as the antimony in the alloy increases. These two curves, indicating the freezing out of Sb and Pb respectively, intersect at the point *r*, or Sb, 13 per cent., at 247° C., the eutectic alloy, which is composed of a mechanical mixture of Pb and Sb due to their simultaneous freezing.

Thus alloys containing Sb from 100 to 13 per cent. are composed of crystals of Sb in an increasing amount of the eutectic or ground-mass, a type of structure which is shown in Figs. 6

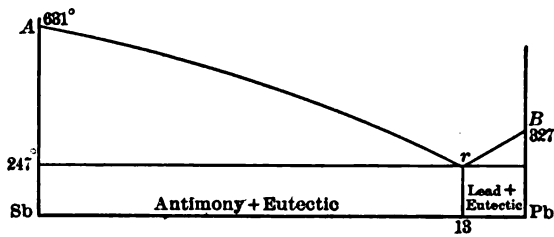

FIG. 1.—THE LEAD-ANTIMONY DIAGRAM.

and 8. At Sb, 13 per cent., the alloy is all eutectic and may be represented by Fig. 7. Alloys of Sb from 0 to 13 per cent. are composed of crystals of Pb in dendritic form set in an increasing amount of the eutectic. Fig. 9 represents this type.

In short, above the curve *ArB* the alloys are completely liquid, below the horizontal line at 247° C. everything is solid. In the left triangle, *Ar*—247° C., are crystals of Sb in a liquid alloy; in the right triangle crystals of Pb in a liquid.

The Pb-Sb alloys are used as bearing-metal, type-metal, hard lead, etc.

2. *Lead-Tin.*

The curve for Pb-Sn is similar to that of Pb-Sb. As determined by Kupffer[18] the eutectic contains Pb, 37 per cent., and freezes at 182° C. Charpy[14] gives Pb, 38 per cent., and Sn, 62 per cent. Roberts-Austen[19] gave the point at Sn, 68 per cent., and 180° C., and published curves showing

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tensile strength and extensibility. Tucker and Rosenhain [20] found the exact point to be Sn, 63 per cent., freezing at 182.5° C.

Thus alloys with Sn less than 63 per cent. are composed of crystals of Pb set in the eutectic: those with Sn more than 63 per cent. are composed of crystals of Sn set in the eutectic, Fig. 11. Solid Pb holds some Sn in solution, the amount varying with the rate of freezing. Rosenhain and Tucker were able to get 16 per cent. in solution by long annealing. As ordinarily made, however, the amount is very much less. Changes in the solid also occur.

The Pb-Sn alloys are used as solders, pewter, in the manufacture of toys, etc.

3. *Tin-Antimony.*

From a microscopic study of Sn-Sb alloys Charpy [21] found a definite compound containing Sb, about 50 per cent., which "formed isomorphous mixtures with Sb." He found (incorrectly) a eutectic at about 10 per cent. of antimony.

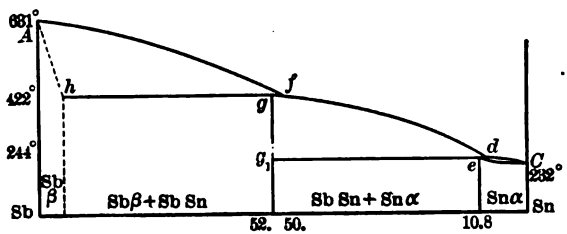

FIG. 2.—THE TIN-ANTIMONY DIAGRAM.

Stead [22] showed that with Sb up to 7.5 per cent. the alloys consist of a solid solution; above this point cubes of a compound Sb-Sn separate out when the alloy freezes. At the Sb end of the series crystals of Sb are the first to form.

The constitution of these alloys was not understood till Reinders [23] published his thermal diagram. Later Gallagher [24] modified the curve and showed a thermal transformation in the cubes (Sb-Sn of Stead) at about 315° C.

Williams [25] gave a simpler curve and omitted the transformation at 315° C. The main points of these three curves are the same, however, and Fig. 2 is taken from them, somewhat simplified, to explain the main changes.

The addition of Sn to Sb lowers the freezing-point pro-

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gressively. The curve shows two inflections, at f and d, with two horizontals, fgh and deg_1. There is no true eutectic, as pure Sn freezes lower than any of the alloys of the series. Such a curve shows that above $AfdC$ all is liquid: below $Ahgg_1eC$ all is solid. Between these two curves we have a mixture of solid and liquid. It also shows that at 422° C. the alloys with Sb from 90 to 50 per cent. show a reaction whereby the solid corresponding to h reacts with the liquid corresponding to f to form solid g. Now h is Sb containing Sn, 10 per cent., in solid solution, Sbβ for short; g corresponds to SbSn. The liquid f contains about 50 per cent. Similarly, at 244° C. we have the reaction of g_1 or Sb-Sn, with the liquid d (8 Sb) to form the solid e containing Sb, 10 per cent. (a solid solution of tin with Sb, 10 per cent., or *Sn* α for short). Then with Sb from 100 to 90 per cent. we have Sbβ. From 90 to 52 per cent. we have Sbβ surrounded by a ground-mass of Sb-Sn. From 52 to 10 per cent. we have Sb-Sn in a ground-mass of Sn α, Fig. 10, while from 10 to 0 per cent. we have Sn α, which is Sn containing Sb in solid solution.[26]

For the sake of simplicity the change in the Sb-Sn at 315° C. has been omitted, also the fact that Sb-Sn probably is capable of holding Sn (some 2 per cent.) in solid solution. In addition, while Sb dissolves Sn up to 10 per cent. at 422° C., the amount at ordinary temperature is much less.

The Sn-Sb alloys have been used as the basis of Britannia metal, for anti-friction alloys, for processes of engraving, etc.

4. *Ternary Alloys of Lead, Antimony, and Tin.*

The ternary diagram has been worked out by R. Loebe[27] and by Campbell and Elder.[28, 29]

The simplest method of representing a series of ternary alloys is by means of an equilateral triangle. In Fig. 3 the corners of the triangle represent the pure metals A, B, and C. The binary alloys of A and B are situated in the line AB, which is a projection of Fig. 1. Similarly, the binary alloys of B and C are represented by the line BC, and the alloys of A and C by the line AC, which is a projection of Fig. 2. An alloy of all three metals is represented by a point within the triangle, for the sum of the perpendicular distance from any point within the triangle to the three sides is constant and is taken as equal

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to 100. Take the alloy represented by O in Fig. 3: the percentage of A is given by the perpendicular from O to BC, the percentage of C by the perpendicular from O to AB, while the percentage of B is given by the perpendicular from O to AC. Hence, if we wish to indicate temperature, the scale must be perpendicular to the plane of the triangle; in other words, we must construct a solid figure.

The simplest case is where each pair of metals has a curve similar to Fig. 1. That is to say, the pairs are completely soluble in the liquid state and insoluble in the solid. Charpy[11] has worked out the Pb-Sn-Bi diagram and Stoffel those for Pb-Sn-Cd and Bi-Sn-Cd.

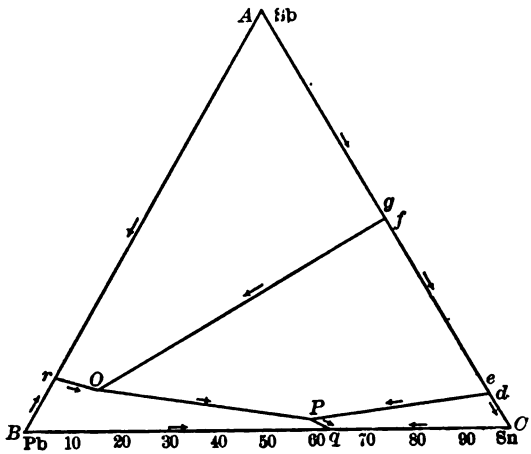

FIG. 3.—THE LEAD-TIN-ANTIMONY DIAGRAM.

More complicated cases have been worked out by Geer[12] and by Sahmen and Vegesack,[13] giving the theory. The latter show the case where the pairs AB and BC form eutectics and have curves represented by Fig. 1, while the pair AC shows no eutectic, but a reaction between the solid h and the liquid f to form the solid g (Fig. 2). Then at O (in Fig. 3) there is a similar reaction whereby solid h' and liquid O react, with the formation of solid g' and B.

In the case of Pb, Sn, and Sb, there are two pairs represented by Fig. 1, and the third by Fig. 2, where there are two reactions. Fig. 3 gives such a ternary diagram, and the following is a brief description of the changes that take place when an alloy solidifies: A represents Sb, B is Pb, and C is Sn. Then

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r is the eutectic of Pb and Sb, with Sb, 13 per cent., freezing at 247° C. Again, q is the lead-tin eutectic at 63 per cent. of tin, freezing at 182° C. On the side AC the point g represents Sb-Sn, f the reaction-point at 422° (f in Fig. 2), d the reaction-point at 244° (d in Fig. 2), and e is Sb, 10 per cent., or saturated Sn α.

In the area $ArOf$ the first solid to form is Sb (with a little Sn in solid solution). In the area $BrOPq$, the Pb crystallizes out first. In $fOPd$, Sb-Sn forms first; while in $CdPq$, the Sn crystallizes out first, with more or less Sb in solid solution (Sn α).

Assuming, for the sake of simplicity, that pure Sb crystallizes out, in the triangle AOr an alloy will deposit crystals of Sb on freezing until the composition of the liquid reaches a point on the line rO. Then we shall have Pb and Sb freezing out side by side, and the liquid composition sliding down the line rO till it reaches the reaction-point, O. At this point occurs the reaction, Sb crystals + liquid O forming Sb-Sn + Pb. If the liquid is used up before the Sb, the alloy goes solid at O. If not, when the Sb has disappeared, the liquid follows the line OP, and the Sb-Sn and Pb separate out side by side till the alloy finally is all solid.

In the area BOr, the first to freeze are crystals of Pb, and the liquid reaches the line rO, the further changes being as above.

In the area AOf, the Sb crystals freeze out, and the liquid reaches the line fO, and follows down it towards O. This change in composition of the liquid is brought about through a reaction between the solid Sb and the liquid to form Sb-Sn. On reaching O, as before, we have the reaction above mentioned, but if the Sb is all gone before the liquid reaches O, further solidification occurs as in the next case.

In the area fPO, the Sb-Sn crystallizes, and the liquid reaches the line OP and follows it, due to the simultaneous freezing of Sb-Sn and Pb. On reaching the point P, we have another reaction, similar to the one at O. Here Sb-Sn + liquid P react to form solid Sn α + Pb. As before, if all of the Sb-Sn is used up before the liquid P, further freezing follows the line Pq, and the alloy finally becomes solid.

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In the area *BOP*, the Pb freezes out first, and the liquid, on reaching line *OP*, follows it, as above.

In the area *BPq*, the Pb crystallizes out first, and on the liquid reaching the line *Pq*, the Pb and Sn (Sn *a*) freeze out side by side till *q* is reached, the binary eutectic of Pb and Sn, also the eutectic of the system.

In the area *fPd* crystals of Sb-Sn are the first to form. When the liquid reaches the line *dP* this stops and we get the reaction Sb-Sn + liquid = Sn *a*, which continues as the liquid follows the curve *dP* to *P*. If the liquid uses up all of the Sb-Sn before *P* is reached, further freezing takes place as in the next case. If not, then at *P* occurs the reaction Sb-Sn + liquid *P* = Sn *a* + Pb and the alloy finally becomes solid.

In the area *CdPq*, the tin solid solution (Sn *a*) first freezes out and then the liquid reaches the line *Pq* as before.

In short, the line *rO*, the boundary between the Sb area and the Pb area, represents the simultaneous freezing of Pb and Sb. The line *OP* is the boundary between the Pb and the Sb-Sn areas and represents the simultaneous freezing of Pb and Sb-Sn; while *Pq* is the boundary between Pb and Sn *a* and represents the simultaneous freezing of these two. All of these give structures resembling that of a true eutectic.

On the other hand, the boundary-line *fO* represents the reaction whereby Sb-Sn forms at the expense of the Sb and *dP* represents the reaction whereby Sn *a* forms at the expense of Sb-Sn. These two points are:

(1) *O* at, Pb, 80; Sn, 10; Sb, 10 per cent., at 245° C.
(2) *P* at, Pb, 40; Sn, 57.5; Sb, 2.5 per cent., at 189° C.

5. *Microstructure.*

In the preparation of sections for the microscope the pieces must be sawed through vertically because there is often a marked difference between the top and the bottom of the specimen. This is due to the fact that both the Sb and the Sb-Sn crystals are much lighter than the liquid out of which they freeze and they therefore tend to float to the surface of the melt. After cutting with a saw the surface of the section must be carefully filed to take off all of the surface altered in the process of cutting. If this is not done the structure will be broken up and

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obscure. After polishing in the usual way the final polish on a rouge-board must be continued longer than usual till a good surface is obtained. As an etching reagent 2 per cent. of nitric acid in alcohol is very good.

With small melts cooled in the air the structure shows that equilibrium is not always established. Alloys in that part of the area *ArOf* the Sb of which ought all to have disappeared, due to the reaction on *fO* or at *O*, often show undissolved cores of Sb. Similarly with Sb-Sn in the area *fOPd*. With very slow cooling, of course, conditions would have brought about more complete equilibrium.

The microstructure of the series is typically shown in the following photographs:

Fig. 6. Pb, 65; Sb, 25; Sn, 10 (× 45). The white crystals are Sb set in a ground-mass which freezes at *O*, a mixture of Sb-Sn and Pb (Fig. 7).

Fig. 7. Pb, 78; Sb, 11; Sn, 11 (× 90). Fig. 8 (the same × 45) occurs very near the point *O*. Fig. 8 is the top of the alloy and shows the layer of Sb-Sn cubes which have floated to the surface. The structure of the ground-mass is well shown in Fig. 7 and is a mixture of Pb and Sb-Sn.

Fig. 9. Pb, 85; Sb, 10; Sn, 5 (× 56). The black dendrites are Pb surrounded by the same ground-mass as before.

Fig. 10. Pb, 20; Sb, 10; Sn, 70 (×45). White cubes of Sb-Sn set in a background of Sn α, which separates out along *dP*, and a dark ground-mass of Pb-Sn α which froze at *P* or on *Pq*.

Fig. 11. Pb, 37.5; Sb, 2.5; Sn, 60 (× 55). The dendrites are Sn (Sn α) surrounded by the Pb-Sn α ground-mass as above.

III. Alloys of Tin, Antimony, and Copper.

These alloys find extensive use as bearing-metal, Britannia metal, etc. Table II. gives a number of typical analyses from various sources and shows that usually Sn exceeds 70, Sb is less than 20, and Cu less than 10 per cent. Charpy [14] examined about 20 alloys for microstructure and compressive strength. Before discussing the ternary alloys a word or two must be said of the Cu-Sn series.

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TABLE II.—*Composition of Sn-Sb-Cu Alloys.*

No.	Name. Metal.	Sn.	Sb.	Cu.		Authority,
		P.C.	P.C.	P.C.	P. C.	
1.	English Britannia	94	5	1		Law.
2.	Bearing	91	4.5	4.5		Private notes.
3.	English Britannia—sheet	90.6	7.8	1.5		Brannt.
4.	English Britannia—cast	90.6	9.2	0.2		Law.
5.	Bearing	90	6	4		Private notes.
6.	Bearing—Russian R. R.	90	8	2		Thurston.
7.	English Britannia	90	6	2	Bi, 2	Law.
8.	English Britannia	90	7	3		Law.
9.	Bearing	89.3	8.9	1.8		Thurston.
10.	Pewter	89.3	7	1.8	Pb, 1.8	Brannt.
11.	Bearing	88.9	7.4	3.7		Karmarsch. Bolley.
12.	Bearing	88.8	7.5	3.7		Law. Thurston.
13.	Queen's metal	88.5	7.1	3.5	Zn, 0.9	Law.
14.	Queen's metal	88.5	7	3.5	Bi, 1.0	Law.
15.	Bearing	87	7	6		Hiorns.
16.	English Britannia	85.5	9.7	1.8	Zn, 3	Law.
17.	Bearing—heavy	85	7.5	7.5		Private notes.
18.	Jacoby metal	85	10	5		Ledebur. Brannt.
19.	German Britannia	84	9	2	Zn, 5	Law. Brannt.
20.	French car-bearings	83.3	11.1	5.5		Charpy.
21.	Bearing	83.3	8.3	8.3		Private notes.
22.	Bearing—German R. R.	83	11	6		Ledebur.
23.	Bearing—valve-rods, etc	82	10	8		Law.
24.	Bearing—French R. R.	82	12	6		Ledebur.
25.	Britannia (Baumgartel)	81.9	16.3	1.8		Bolley. Brannt.
26.	Bearing—Swiss R. R.	80	10	10		Thurston.
27.	Ashberry metal	80	14	2	Zn, 1	Law.
28.	Ashberry metal	79	15	3	Zn, 2	Law.
29.	Britannia. Ashberry	77.8	19.4	2.8		Thurston. Bolley.
30.	Britannia. Ashberry	77.9	19.4	.....	Zn, 2.8	Roberts-Austen.
31.	Bearing—English	76.7	18.5	7.8		Brannt. Bolley.
32.	Bearing—German	76	17	7		Brannt.
33.	Bearing	73	18	9		Thurston.
34.	Bearing	72	26	2		Thurston. Bolley.
35.	German Britannia	72	24	4		Brannt. Law.
36.	Bearing (Karmarsch)	71.4	7.2	21.4		Thurston. Bolley.
37.	Bearing—Valve packing	71	24	5		Thurston.
38.	Bearing (Karmarsch)	70.7	19.7	9.5		Thurston. Bolley.
39.	Minofor (Britannia)	68.5	18.2	3.3	Zn, 10	Brannt. Law.
40.	Bearing—G. W. R. (England)	67	11	22		Thurston.
41.	Bearing—French R. R.	67	22	11		Law. Charpy.
42.	Dewrance metal (locomotive)	33.3	44.5	22.2		Charpy. Hiorns.

1. *Copper and Tin.*

In Fig. 4 is given a simplified diagram. Some of the changes have been omitted (*e.g*, that which occurs in the alloys between *c* and *C* at 740° C.) because the various researches, notably by Heycock and Neville, Shepherd and Blough,[30] Giolotti and Tavanti,[31] do not agree, and these points are still unsettled. The freezing-point curve, or liquidus, is *ABCDEFG*. The solidus, or end of freezing, is given by *AabcdxnefFg*. In alloys with Sn from 0 to 38.2 per cent. (Cu_3Sn) the curves below the solidus represent changes in the solid alloy.

Considering the Cu-Sn thermal diagram in small parts it at once becomes simple, being merely compounded of a number of simple curves.

The Two Solid Solutions, Alpha and Beta.

The part of the diagram embraced by *ABC—Aabc* and just

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below it, shows that the conditions are similar to that represented by Fig. 2.

In freezing, a solid solution rich in Cu (alpha or the solid *I*) separates out when the temperature reaches the curve *AB*. This enriches the liquid in Sn, and at 795° C. there are dendrites of alpha, saturated with Sn of composition *a*, and a liquid of composition *B*. Then occurs a reaction: solid solution alpha (*a*) + liquid B = solid solution beta (*b*). This reaction continues till all of the alpha or all of the liquid is used up and the alloy

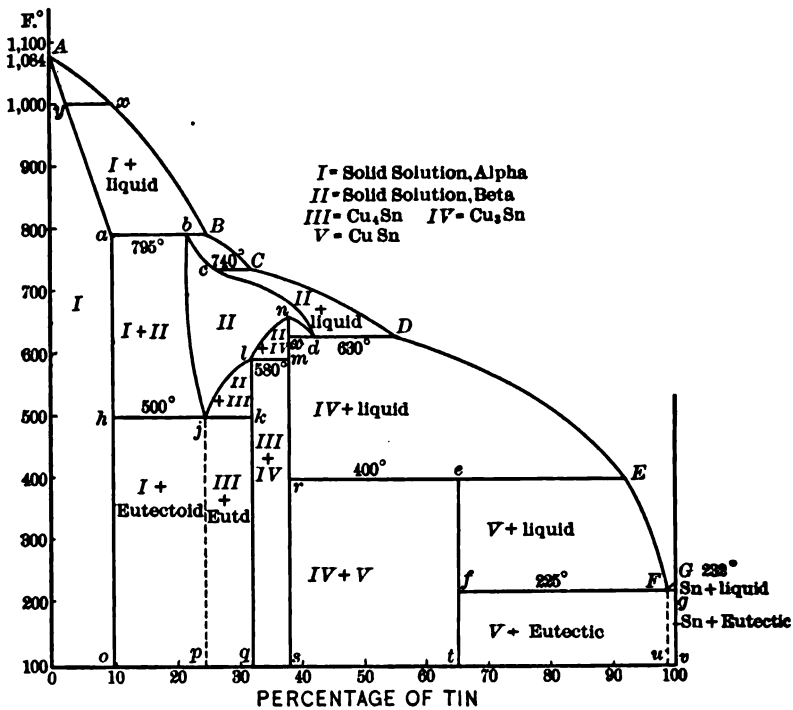

FIG. 4.—THE COPPER-TIN DIAGRAM.

becomes solid. In the area marked by the solid *I* is the solid solution rich in copper, or alpha. In the triangular area marked by the solid *II* is nothing but the beta solid solution, while between these two areas is *I* + *II*, or the mixture of alpha and beta (see Figs. 18 and 24).

But this beta solid solution, which contains Sn, 22 per cent., at 795° C., becomes enriched in Sn as the temperature falls, and this it does by following down the curve *bj* and precipitat-

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ing some alpha. On arriving at *j*, the eutectoid point at 500° C., it suddenly breaks up into a mechanical mixture of alpha and another solid which we will call Cu_4Sn (although it may be merely a solid solution and not a compound). This mechanical mixture is the eutectoid and corresponds to pearlite in steel.

Beta Alone.

In alloys between *b* and *p* (Sn, from 22 to 25 per cent.) the beta solution in the area *II* in cooling reaches the curve *bj* and alpha separates out, enriching beta in tin until at 500° C. it corresponds to *j*, or Sn, 25 per cent., when it splits up into the eutectoid as before.

In alloys between *p* and *q*, or Sn, 25 and 31.8 per cent. respectively, the beta solution in the area *II*, on cooling down, reaches the curve *lj*, and *III*, or Cu_4Sn, starts to precipitate. Thus in area *ljk* are grains of beta surrounded by a network of Cu_4Sn, just as in high-carbon steel the solid solution (austenite) can occur surrounded by iron carbide (cementite). At 500° C., or *hjk* line, the remaining beta of composition *j*, or Sn, 25 per cent., splits up into the eutectoid as before.

Therefore into the normal state we have:

Sn from 0 to 10 per cent. (alpha solid solution). Sheet-bronze to gun-metal.

Sn from 10 to 25 per cent. (alpha grains set in eutectoid, Figs. 24 and 19). Gun-metal to bell-metal.

Sn from 25 to 31.8 per cent. (Cu_4Sn and eutectoid, see Fig. 12).

Sn 31.8 per cent. (pure Cu_4Sn). Speculum-metal.

Limit of the Alpha Solid Solution.

The amount of Sn that Cu holds in solid solution depends on the rate of freezing and cooling. In an ordinary casting, when Sn exceeds 6 per cent., traces of the eutectoid can be seen. At Sn, 10 per cent. (a gun-metal), the structure closely resembles Fig. 24, the euctectoid being quite noticeable. In these alloys, however, the alpha grains are not saturated. Long annealing gives us homogeneous alpha grains, up to Sn, 12 or 13 per cent. Thus sheet gun-metal is all alpha.

The eutectoid is hard, sonorous, but somewhat brittle. Thus

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for small bells we can use almost the eutectoid alloy, while for larger ones we must have more of the alpha grains, which decrease the brittleness and lower the tone. Fig. 19 might represent the structure of ordinary bell-metal.

The compound Cu_4Sn is very brittle, but takes a high polish and is used for mirrors in optical work.

The Compounds Cu_4Sn to Cu_3Sn, or Sn, 31.8 to 38.2 Per Cent.

Taking the area *II* above the curve *ln*, the beta solid solution, on cooling down, reaches *nl*, and the compound Cu_3Sn, or *IV*, precipitates out. Thus in the area *mln* we have a mixture of beta and Cu_3Sn. Now at 580° C., or *lm*, the remaining beta changes into the compound Cu_4Sn, so that below 580° C. these alloys consist of Cu_3Sn and Cu_4Sn, *i. e.*, *III* plus *IV*. They are very brittle and not commercially useful.

Alloys Rich in Tin.

The alloys with Sn in excess of 38.2 per cent. are all brittle and of no commercial importance until those containing about 85 per cent. are reached. This part of the curve shows two horizontals, *Dx* and *Eer*, besides the eutectic line *fFg*. At 630° C. the line *Ddx* gives the temperature of the reaction solid solution *II* + solid *IV*, or Cu_3Sn + liquid *D*. Similarly, the line *Eer* at 400° C. represents the reaction: *IV* (Cu_3Sn) + liquid *E* = *V* (CuSn). Theoretically this reaction ought to be complete, and in alloys with Sn from 38.2 to 65 per cent. the liquid is all used up before the Cu_3Sn, and therefore the kernels of Cu_3Sn are surrounded by a ground-mass of hard white Cu-Sn, or *IV* + *V*. In practice, however, the Cu-Sn forms envelopes round the Cu_3Sn and the reaction is not complete, so in the area *IV* + *V* occurs Cu_3Sn + Cu-Sn and some eutectic, while in alloys with Sn from 65 to 92 per cent., *e* to *E*, are more or less cores of Cu_3Sn inside the Cu-Sn set in the excess eutectic, Fig. 15. Theoretically all of the Cu_3Sn should be used up (and is, on very slow cooling) before the liquid and the resulting alloys should be Cu-Sn (*V*) + eutectic.

On the right hand side of the diagram is the *V* curve *EFG;* this is similar to that of Fig. 1, for *EF* denotes the freezing-out of Cu-Sn in laths and needles, *FG* the freezing of crystals of Sn in dendrites, while the horizontal *gFf* is the freezing of the

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eutectic of Sn and Cu-Sn. See Figs. 14, 13, and 12. The Cu-Sn needles, skeleton stars, etc., are quite characteristic, and together with the CuSn-Sn eutectic enter into the constitution of tin-rich babbitt metals.

The horizontal *Een* illustrates the case of the compound, which dissociates before reaching the melting-point; thus, on heating, Cu-Sn, or *V*, splits up at 400° C. into Cu_3Sn (*IV*) + liquid *E*.

The eutectic point *F* is Cu, 1 per cent., and 225° C. (Fig. 13). In the alloys with less than Cu, 1 per cent., there are grains and dendrites of Sn, surrounded by the eutectic (Fig. 12). Thus, in the useful alloys are:

Cu from 15 to 8 per cent. Cores of Cu_3Sn, envelopes, and needles of Cu-Sn set in the eutectic. Fig. 15.

Cu from 8 to 1 per cent. Cu-Sn needles, stars, etc., in the eutectic. Fig. 14.

Cu from 1 to 0 per cent. Dendrites and grains of tin, surrounded by the eutectic of Sn and Cu-Sn. Fig. 12.

2. *Ternary Alloys of Tin, Antimony, and Copper.*

In Fig. 5 is given an approximate diagram of Sb up to 30 per cent., and Cu, 30 per cent., worked out from microstructure. The changes which take place are similar to those of the Pb-Sb-Sn series.

The line *cq* represents the reaction between Cu_3Sn and the liquid to form Cu-Sn. The line *do* is the reaction between Sb-Sn and the liquid to form Sn *a*. The line *po* is the simultaneous freezing of Sb-Sn and Cu-Sn, while *ob* is the simultaneous freezing of Sn *a* and Cu-Sn, giving a eutectic-like structure, as in Fig. 13.

The point *O* is approximately Cu, 1; Sb, 7.5; Sn, 91.5 per cent., and represents the reaction: Sb-Sn + liquid *o* = Sn *a* + Cu-Sn.

Any alloy on the left of *qc* will freeze out Cu_3Sn, and when the liquid reaches the line *cq*, the reaction to form Cu-Sn occurs. Now in ordinary cooling the Cu_3Sn is soon covered by Cu-Sn (Fig. 15), and the reaction stops, while Cu-Sn continues to freeze out normally from the liquid. The further structure of the alloy depends on whether the liquid reaches the line *po* or *ob*. Suppose the liquid reaches *po*, then there is a

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simultaneous separation of Cu-Sn and Sb-Sn, following *po*, till at *O* occurs the reaction to form Sn α + Cu-Sn. Again, the reaction is not completed, and the composition of the liquid follows *ob* till the whole is solid. This is shown in Fig. 17: Sn, 67; Sb, 11; Cu, 22 (× 45). The dark cores to the plate-like crystals are Cu_3Sn. The white envelopes and dots are Cu-Sn, the white cubes are Sb-Sn, and the ground-mass is a mechanical mixture of Cu-Sn and Sn α (see Fig. 13), which freezes along *ob*.

The case where the liquid reaches *ob* is simpler, for it freezes along *ob* as before and the alloy shows no cubes of Sb-Sn. Fig. 15, Sn, 70; Sb, 5; Cu, 25 (× 45), shows the dark cores of Cu_3Sn surrounded by the white envelopes of Cu-Sn, in a ground-mass which freezes along *ob*.

Coming next to those alloys the composition of which falls within *pobcq*, the first thing to freeze out is Cu-Sn, thus enriching the liquid in Sn and Sb, until it arrives at *po* or *ob*. The further changes are of two kinds, as above. Fig. 16, Sn, 83.5; Sb, 11; Cu, 5.5 (× 45), started to freeze by separating needles and skeleton crystals of Cu-Sn. The liquid finally reached the line *po* and then occurred a simultaneous freezing of cubes of Sb-Sn and needles of Cu-Sn until the point *O* was reached. Then followed reaction and solidification along *ob* as before. Fig. 14, Sn, 90; Sb, 4.5; Cu, 4.5 (× 50), began to freeze out Cu-Sn as before, but the liquid reached the line *ob* and followed along it till solid. Therefore no cubes appear.

Any alloy in the area *pode* first freezes out cubes of Sb-Sn. These are lighter than the liquid out of which they freeze, and float to the surface, as shown in Fig. 8. The liquid may reach *op* or *od*. If *op*, then Sb-Sn cubes and Cu-Sn needles crystallize out together, and freezing continues, as already described. If *od*, the cubes tend to react with the liquid to form Sn α till *o* is reached. This reaction is never completed in ordinary cooling, and further freezing follows *ob*, as before.

The reason why the cubes in Fig. 16 have not floated to the surface is because they are held down by a skeleton of Cu-Sn crystals, which froze before or with them. This is well shown in Fig. 17.

Alloys on the line *ob* are represented by Fig. 13: Sn, 94; Sb, 5; Cu, 1 (× 50). Here we see Sn α and Cu-Sn separating simultaneously, and we get a pseudo-eutectic structure.

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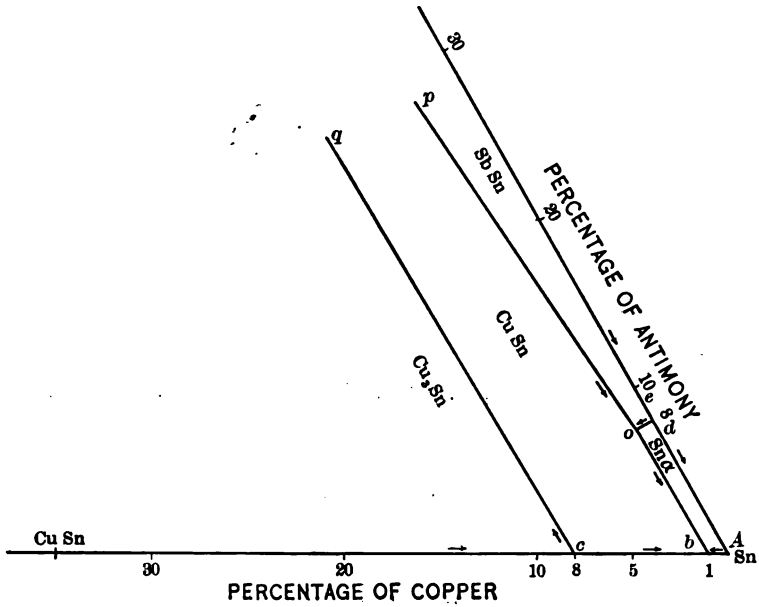

FIG. 5.—PART OF THE TIN-ANTIMONY-COPPER DIAGRAM.

FIG. 6.—PB, 65 ; SB, 25 ; SN, 10. × 45.

FIG. 7.—PB, 78 ; SB, 11 ; SN, 11. × 90.

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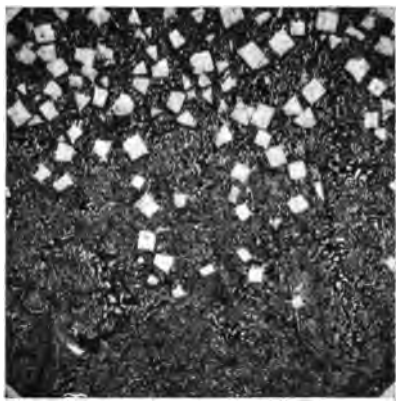

FIG. 8.—Pb, 78 ; Sb, 11 ; Sn, 11. × 45.

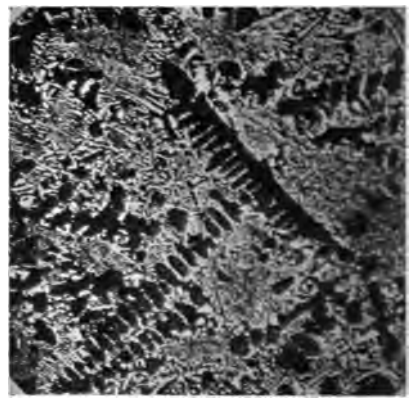

FIG. 9.—Pb, 85 ; Sb, 10 ; Sn, 5. × 56.

FIG. 10.—Pb, 20 ; Sb, 10 ; Sn, 70. × 45.

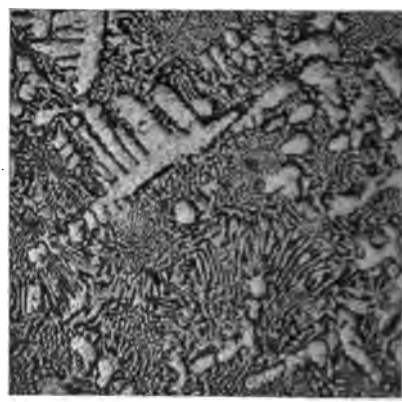

FIG. 11.—Pb, 37.5 ; Sb, 2.5 ; Sn, 60. × 55.

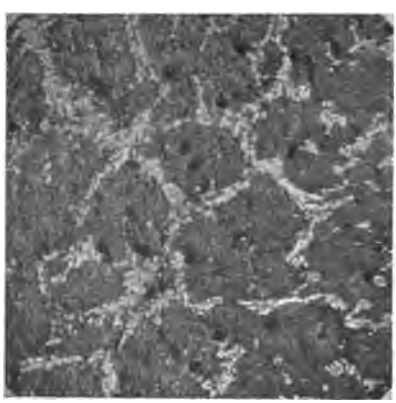

FIG. 12.—Sn, 94.5 ; Sb, 5 ; Cu, 0.5 × 45.

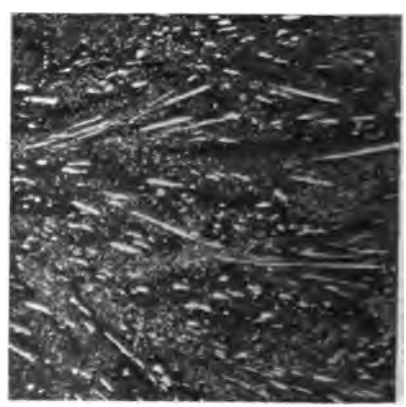

FIG. 13.—Sn, 94 ; Sb, 5 ; Cu, 1. × 50.

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FIG. 14.—Sn, 90 ; Sb, 4.5 ; Cu, 4.5. × 50.

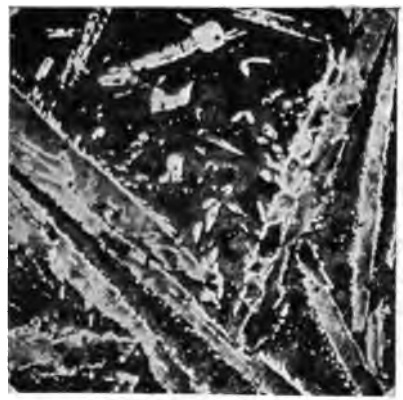

FIG. 15.—Sn, 70 ; Sb, 5 ; Cu, 25. × 45.

FIG. 16.—Sn, 83.5 ; Sb, 11 ; Cu, 5.5. × 45.

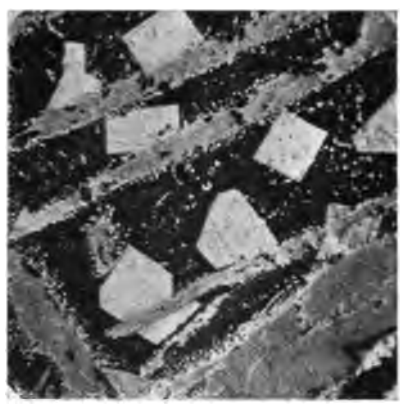

FIG. 17.—Sn, 67 ; Sb, 11 ; Cu, 22. × 45.

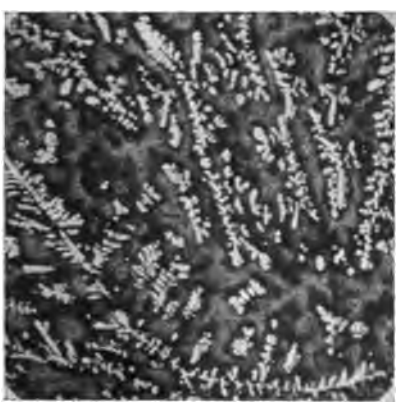

FIG. 18.—Zn, 96 ; Cu, 4. × 45.

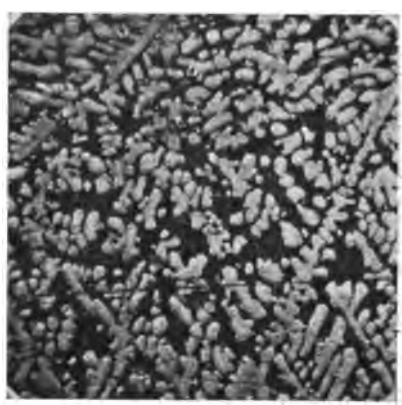

FIG. 19.—Zn, 85 ; Cu, 10 ; Al, 5. × 55.

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FIG. 20.—Zn, 85; Cu, 5; Sn, 10. × 45.

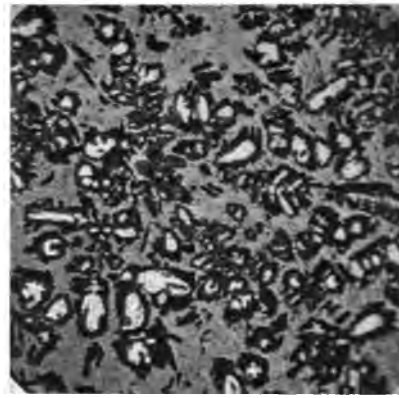

FIG. 21.—Zn, 35; Sn, 60; Cu, 5. × 45.

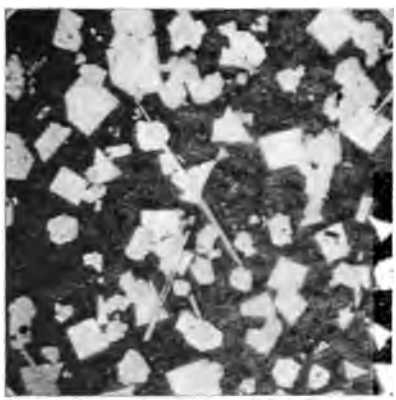

FIG. 22.—Sn, 78; Sb, 19; Zn, 3. × 45.

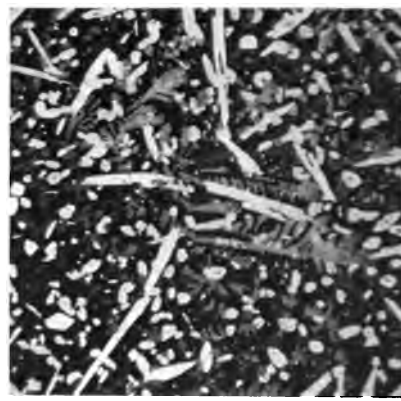

FIG. 23.—Zn, 85; Sb, 10; Cu, 5. × 45.

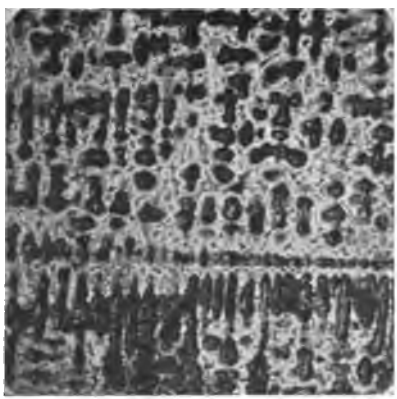

FIG. 24.—Cu, 88; Sn, 10; Zn, 2. × 45.

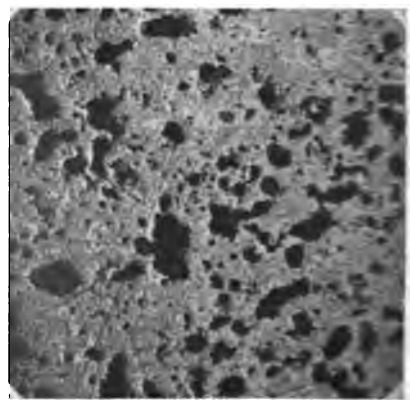

FIG. 25.—Cu, 62; Sn, 11; Pb, 27. × 65.

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Lastly, alloys within the area *bodA* consist of dendrites and grains of Sn *a* set in this pseudo-eutectic, which freezes along *ob*. Fig. 12 shows the alloy Sn, 94.5; Sb, 5; Cu, 0.5 (× 95). The dark etching grains are the Sn *a* surrounded by the lighter ground-mass, a mixture of Sn *a* and Cu-Sn.

From the above, it will be seen that in this series most of the alloys are not in a state of equilibrium, because the various reactions are incomplete. As the copper-content is increased the amount of Cu-Sn increases, till, finally, Cu_3Sn is present. This increase in Cu increases the hardness (and brittleness) and decreases the plasticity proportionately. Similarly, an increase in Sb increases the amount of the hard cubes of Sb-Sn, with similar results.

IV. Zinc-Base Alloys.

Alloys rich in Zn have an extensive use as bearing-metals. Most of them contain Cu as well as Sn. Table III. gives the analyses of a number of alloys which have been used or advocated.

Table III.—*Composition of Zinc-Base Alloys.*

No.	Name. Metal.	Zn.	Cu.	Sn.	Sb.		Authority.
		P.C.	P.C.	P.C.	P.C.	P.C.	
1.	Hamilton metal	93.4	3.5	.....	1.5	Pb, 3.1	Thurston. Bolley.
2.	Bearing—hard	90	7	1.5	1.5	.....	Brannt.
3.	Ehrhardt's metal	89	4	4	.....	Pb, 3	Thurston. Bolley
4.	Bearing	88	8	2	2	.....	Brannt.
5.	Salge metal	85.5	4	9.9	.....	Pb, 1.1	Dudley.
6.	Lumen	86	10	.....	.....	Al, 4	Metal Industry.
7.	Lumen	85	10	.....	.....	Al, 5	Notes.
8.	Bearing	85	5	.....	10	.....	Ledebur.
9.	Bearing	85	5	10	.....	.....	Notes.
10.	Biddery metal—Heine's	84.3	11.4	1.4	.....	Pb, 2.9.	Bolley. Brannt.
11.	Pierrot metal—Beugnot	83.3	8.3	7.6	3.5	Pb, 3	Ledebur.
12.	Fenton's alloy	80	6	14	.....	.....	Law.
13.	Fenton's alloy	80	8.5	.....	14.5	.....	Thurston. Bolley.
14.	Bearing	77	5.5	17.5	.....	.....	Ledebur.
15.	English white metal	76.2	5.6	17.5	.....	Pb, 0.7	Brannt.
16.	Vaucher's alloy	75	.....	18	2.5	Pb, 4.5	Hiorns.
17.	For pump-cocks	72	7	21	.....	.....	Bolley.
18.	For propeller-bushing	69	5	26	.....	.....	Roberts-Austen.
19.	Babbitt metal	69	5	19	3	Pb, 5	Ledebur.
20.	Bearing—English	67.7	7.4	14.9	.....	.....	Thurston. Bolley.
21.	Bearing	66.5	4.2	29.3	.....	.....	Thurston. Bolley.
22.	Dunnlevic & Jones metal	52	1.6	40	0.4	.....	Charpy.
23.	Heavy axle	47	1	38	6	Pb, 4	Brannt.
24.	Heavy axle	40	3	15	.....	Pb, 3.2	Brannt.
25.	Parsons white brass	30	5	65	.....	.....	Notes.

The binary alloys of Zn and Cu show at the Zn end of the series (Shepherd,[8] Tafel [32]) two solid solutions, ϵ, with Cu,14 per cent., and η, with Cu, 2.5 per cent. There is no eutectic, but a reaction-point, as in the alloys rich in Sn in Fig. 2. Thus, Fig. 18 (× 45) is the alloy with Zn, 96; Cu, 4 per cent. The white

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dendrites are the ϵ with Cu, 14 per cent., and the dark etching ground-mass is η with Cu, 2.5 per cent., in solid solution. Now, most of the zinc-base bearing-metals show dendrites of ϵ set in a ground-mass, which varies with the third metal.

1. *Zinc and Tin.*

The Zn-Sn alloys can be shown by Fig. 1. The eutectic occurs at Zn, 8 per cent., and 204° C. As the Zn end of the Cu-Zn diagram can be represented by the right-hand side of Fig. 2, the zinc-rich corner of the ternary diagram can be explained by the right-hand corner of Fig. 3, if d represents Cu, 2.5 per cent., and q, Sn, 92 per cent., the C would be pure zinc. For example, Fig. 20 is Zn, 85; Cu, 5; Sn, 10 (× 45). The white dendrites are ϵ (Cu, 14; Zn, 86), and crystallized out until the liquid reached the line dP. Following this line there was the reaction between ϵ and the liquid to form η, which continued until at P the rest of the liquid went solid, for ϵ + liquid = η + Sn. The latter can be seen as irregular light lines in the ground-mass.

Again, Fig. 21: Zn, 35; Sn, 60; Cu, 5 (× 45). As before, the white dendrites are ϵ, surrounded by dark envelopes, the η, while the ground-mass is the matrix rich in Sn, which froze at P, as before.

2. *Zinc and Aluminum.*

The metals Zn and Al were thought to form no compound (Shepherd[33]), but Rosenhain and Archbutt[39] show that the eutectic Al, 5 per cent., at 380°, is composed of a mechanical mixture of Zn and Al_2Zn_3.

Then in the alloy shown in Fig. 19, Zn, 85; Cu, 10; Al, 5 (× 55), the white dendrites are ϵ, as before, but the ground-mass, which freezes mainly at the point P (Fig. 2), is composed of η and Al_2Zn_3. Under high powers it is clearly such a mechanical mixture.

3. *Other Alloys of Zinc.*

Two other alloys are given as representing a different form of crystallization. Fig. 23, Zn, 85; Sb, 10; Cu, 5 (× 45), the bright needles are Sb_2Zn_3, set in a complex ground-mass. Fig. 22 is an alloy rich in Sn—Sn, 78; Sb, 19; Zn, 3 (× 45). There are three distinct kinds of crystals: the bright white cubes of

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Sb-Sn, darker cube-like forms, perhaps Sb-Zn, with a few needles of Sb_2Zn_3, the whole set in a ground-mass rich in tin.

To make a general summary of the structure of these white anti-frictional alloys, they are composed of hard crystals or dendrites, such as Sb-Sn, Cu_3Sn, Cu-Sn, Sb_2Zn_3, ϵ, etc., set in a more or less plastic ground-mass.

V. The Bronzes.

For the sake of comparison between the white metals and the bronzes, two micrographs are given, Figs. 24 and 25. The first, Fig. 24, shows the alloy of Cu, 88; Sn, 10; Zn, 2 ($\times$ 45). It is composed of composite dendrites, with dark etching cores, set in a small amount of a harder material, the eutectoid. The structure can be explained by reference to Fig. 4. Neglecting Zn for a moment, an alloy of, say, Cu, 90; Sn, 10, on cooling down to the temperature of the liquidus curve, AB, begins to crystallize at about 1,000 C., or x. The composition of the first solid to form is given by the solidus curve, Aa, and is y. Now, as freezing continues and the temperature falls, the composition of the liquid follows the curve AB, the composition of the solid follows the curve Aa, until, at 798° C., the last trace of liquid of composition B has disappeared, and the solid has now the composition a, or Sn, 10 per cent. This state of affairs only occurs with slow cooling, such as to allow the composition of the solid to change progressively along the curve Aa by diffusion. Under ordinary conditions, however, cooling is not slow enough to maintain a state of equilibrium, and the diffusion does not keep pace with freezing, so at the temperature of the line abB the solid contains less Sn than a, and some liquid B remains. In other words, the dendrites of the solid, or I, are richer in Cu in the center, or cores, and etch differentially, as seen in Fig. 24. At 795° the reaction takes place, solid a + liquid B = solid b, and all of the liquid is used up. Hence, below 795° C. are dendrites of a, surrounded by a small amount of β, or $I + II$. Then, at 500° C., the β splits up into the eutectoid, as already described. Hence, an alloy, when cold, consists of unhomogeneous dendrites of α set in a small amount of the eutectoid, as in Fig. 24. In regard to the Zn, 2 per cent., Cu, on freezing, will retain Zn to 30 per cent. in solid solution, or 38 per cent. at ordinary temperatures. In the ternary Cu-Zn-Sn diagram a triangle

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bounded by Cu to Cu, 90-Sn, 10; and Cu to Cu, 64-Zn, 36, gives the approximate limits of the solid solution α. In the alloys rich in Sn, the excess is the eutectoid, as in Fig. 24, all the Zn being in solid solution in the dendrites of alpha. In alloys rich in Zn, as tobin bronze, naval brass, manganese-bronze, the excess is the beta Cu-Zn solid solution, the Sn being all in solution, presumably in the alpha, and therefore the resulting structure is that of muntz metal.

1. *Phosphor-Bronze.*

These alloys may be divided into low and high phosphorus. In low-phosphorus alloys the phosphorus is completely dissolved in the alpha. In high-phosphorus alloys the compound Cu_3P is present as a "pseudo-eutectic," mixed with the eutectoid of alpha and Cu_4Sn. The formation is similar to that shown in the area *AnOf* in Fig. 3, taking *r* as the Cu-Cu_3P eutectic at 8.27 per cent., and 700° C., according to Heyn and Bauer,[35] and *f*, the reaction-point *B* of Fig. 4, at Sn, 25 per cent., and 795° C. The point *O* is probably 620° C., and Cu, 81; Sn, 14.2; P, 4.8 per cent., from Hudson and Law's work[36] on phosphor-bronze. Thus, in freezing, the first constituent to form is alpha, as in Fig. 24. On reaching the curve *fO* occurs the reaction which forms beta. At *O* the whole mass goes solid by the reaction: alpha + liquid *O* = beta + Cu_3P. Then, at 500° C., the beta splits up into the eutectoid, as in ordinary bronzes. Hence, the structure is composed of alpha dendrites in a ground-mass with a eutectic-like structure, a mixture of Cu_3P, Cu_4Sn, and alpha.

2. *Plastic Bronze.*

The structure of the bronzes rich in Pb is typically shown in Fig. 25: Cu, 62; Sn, 11; Pb, 27 (× 65). The black patches are Pb, the background is bronze (alpha eutectoid). This structure is due to the fact that Pb has only a limited solubility in Cu and in bronze.

In the Cu-Pb alloys, Cu freezes out and enriches the liquid in Pb until at 950° C. it is saturated with Pb, 35 per cent. Further freezing out of Cu causes the precipitation of a second liquid rich in Pb, which freezes independently when the temperature falls after the first liquid has all frozen. This precipitation of the second liquid rich in Pb gives rise to the

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structure given in Fig. 25, the final freezing taking place at 327° C.

The ternary alloys Cu-Sn-Pb have been studied by Giolitti and Marantonio[37] and by Guertler,[38] who found that the saturation-point of Cu, 65; Pb, 35, at 950° C., was reduced by the addition of Sn both in concentration and temperature. Thus Sn, 10 per cent., reduces the point to 885° C. and Pb 23 per cent., whereas Sn, 20 per cent., reduces it to 796° C. and Pb 10 per cent. Again, the addition of Pb to a Cu-Sn alloy (say Sn, 10; Cu, 90) reduces the freezing-point. Cu-Sn alpha begins to freeze out and enriches the liquid in Pb. At the saturation-point the liquid rich in Pb is precipitated as before; while the transition-point, α + liquid = β, is lowered by the addition of Pb, the eutectoid change at 500° C., where the β decomposes, is not lowered thereby, because the composition of the β is not altered by the lead.

In general, then, the structure of ordinary bronze consists of the Cu-Sn solid solution alpha with a small amount of the hard eutectoid filling in the interstitial spaces between the dendrites and grains. Phosphor-bronze is similar, but in addition to the eutectoid Cu_3P is present, with a proportionate lowering of both freezing- and melting-points. Lastly, the plastic bronzes show, in addition to the Cu-Sn solid solution alpha and the euctectoid, more or less patches of Pb, which has reduced the freezing-point and retarded the end of solidification to 327° C., the freezing-point of Pb. The presence of this Pb gives the plastic property.

Appendix.

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The Alundum Extraction-Thimble Used in the Determination of Copper.

BY L. W. BAHNEY, NEW HAVEN, CONN.

(Cleveland Meeting, October, 1912.)

THE photograph, Fig. 1, shows the apparatus a little less than half size, consisting of a filtering-flask fitted with rubber stopper through which passes a bent glass tube, and an extraction-thimble fitted with rubber stopper through which passes a glass tube of 0.25-inch bore. Both tubes are connected by a short piece of rubber tubing.

A section of a thimble is shown in the photograph; the tube extends to within $\frac{1}{8}$ in. of the tapered end.

The object of using the thimble is to remove the acid from the beaker after all the copper has been precipitated. Time is saved, the copper is not exposed to the acid alone, and there are none of the losses attending ordinary filtration. I have accomplished these results by means of a piece of perforated platinum fastened in the end of a 0.25-in. bore glass tube and a filter-mat of asbestos, but after my supply of proper length fiber became exhausted I could not replenish it even after purchasing 14 lots from four different dealers.

The above apparatus may be used to remove at least seven-eighths of a supernatant liquid from a settled precipitate without disturbing the latter.

The application of the thimble is best shown by partly outlining the assay for copper, as follows:

Dilute the acid solution of copper and other sulphates to 150 cc. in a 200-cc. Jena beaker, place on a hot plate, add 2 drops of concentrated HCl, then place a strip of aluminum in the beaker (this may be bent or straight, as desired). Connect the apparatus, as shown in Fig. 1, with a filter-pump having a strong suction. When the copper is precipitated, remove the beaker from the hot plate and insert the extraction-thimble alongside the strip of aluminum. The acid solution will be drawn through the porous tube. Wash the upper end of the aluminum strip with a jet of hot water, wash down the sides of

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the beaker, and add about 25 cc. of hot water to cover the precipitate.

As soon as all the liquid is out of the beaker, disconnect the thimble by slipping the rubber tubing from the bent glass tube of the filtering-flask, wash it with a jet of hot water. If there is no adhering copper, lift from the beaker, add 5 cc. of strong nitric acid to the precipitated copper, and carry out the titration in the usual manner. If the copper is in a finely-divided condition as a precipitate on the aluminum, it will adhere as a black coating on the thimble, but as soon as the thimble is dis-

FIG. 1.—THE ALUNDUM EXTRACTION-THIMBLE AS USED IN THE DETERMINATION OF COPPER.

connected the water inside it will ooze out through the pores, and if the 5 cc. of strong nitric acid be poured on it, the copper will immediately dissolve and the thimble may be washed with a jet of hot water. The filtration is more rapid when the solution is hot.

If the last of the solution in the beaker is not withdrawn as rapidly as the first, tip the beaker at an angle of 45°. With a pump having a strong suction this will not be necessary.

The thimble attached to the flask, as shown above, was used in more than 300 determinations.

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The Sampling of Gold-Bullion.*

BY FREDERIC P. DEWEY, WASHINGTON, D. C.†

(Cleveland Meeting, October, 1912.)

AT the Seventh International Congress of Applied Chemistry I presented a paper,[1] The Assay and Valuation of Gold-Bullion, in which are briefly mentioned a few illustrations of different methods of sampling gold-bullion, particularly cyanide-bars. Since then I have carried on an extensive investigation upon the sampling of gold-bullion in relation to its effects upon the assay-results, in connection with the statement of accounts between the smaller assay-offices of the United States which purchase gold-bullion and the mints where the purchases are re-deposited.

Under the name of gold-bullion are included metals of widely different composition. We may have practically pure metal, 999.75 fine or even more in gold; alloys of gold and silver with only trifling amounts of base metal; alloys of gold and copper with trifling amounts of other metals; ternary alloys of gold, silver, and copper, or gold, silver, and lead; and most complex alloys of from five to eight or more constituents of ordinary occurrence, besides some of the rarer metals occasionally. In any one or all of these classes we may have one or more of the metalloids, which may add to our difficulties.

Of course, with metal 999.75 fine in gold there is only slight opportunity for different samples to differ in fineness, but on such fine metal we expect the assays to agree very closely, and the sampling must be done with the utmost care. When possible, dip- or granulation-samples of the molten metal should be taken; but when it is necessary to sample bars, the chief point to guard against is surface-dirt.

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* This paper was read at the Eighth International Congress of Applied Chemistry, New York, September, 1912, in advance of its appearance in the *Annual Report of the Director of the Mint for the Year* 1911-12, through the kind permission of the Director. It is here published by mutual agreement.

† Assayer, Mint Bureau, U. S. Treasury, Washington, D. C.

[1] *Trans.*, xl., 780 to 797 (1909), and in *Annual Report of the Director of the Mint for the Year* 1908-09, p. 25.

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In the case of four melts of fine gold, which had been carefully sampled by both dips and chips, one of the chip-bars and an independent bar were chipped, and the chips assayed. Table I. summarizes the results.

TABLE I.—*Fine Gold Sampling.*

Original Report. Fineness.	Sample-Bar. 2d Sample. Fineness.	2d Bar. Fineness.
997.9	997.7	997.9
	998.	
997.5	997.4	997.4
	997.3	
997.4	997.4	997.3
	997.4	
997.2	996.9	996.9

In the case of four melts of metal of much higher grade, independent samples were taken, and assayed after the melts had been reported by the assayer. Table II. summarizes the results.

TABLE II.—*Fine Gold Sampling.*

Original Report. Fineness.	Resamples. Fineness.
999.8	999.7
	999.7
999.7	999.7
	999.7
999.7	999.7
	999.6
999.6	999.5
	999.5

Our standard gold (coin-gold) contains 900 parts of gold and 100 of copper. When made from pure metals it does not segregate. Four melts of standard gold were carefully sampled by both dips and chips and reported by the assayer. Subsequently one of the sample-bars and a second bar were chipped, and the chips were assayed. Table III. summarizes the results.

TABLE III.—*Standard Gold Sampling.*

Original Report. Fineness.	Sample-Bar. 2d Sample. Fineness.	2d Bar. Fineness.
900	899.8	899.8
	899.9	
899.9	899.8	899.8
899.8	899.7	899.6
899.7	899.7	899.5
	899.8	

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In the cases already cited the sampling is a very simple matter, and the variations in the results shown may as well be due to the assaying itself as to the sampling. When, however, we come to consider miscellaneous bullion of more complex composition, the sampling assumes greater importance.

From a bar of strictly homogeneous metal, manifestly it would make no difference how the sample was taken, since any part of the metal would represent the whole. Bars that are practically homogeneous are not uncommon; and such bars would not have to be melted for the purpose of sampling, if we could be certain that they were homogeneous. Only two classes of alloys can form homogeneous bars—those that are solid solutions, and the eutectics; and unfortunately there are no characteristics by which the homogeneity of a bar of gold-bullion may be readily established. As a general proposition it is safe to assume that a brittle bar of gold-bullion will not be homogeneous. We may, of course, happen to get a bullion of eutectic composition, which would be both brittle and homogeneous, but eutectics undoubtedly form a very small proportion of the ordinary run of gold-bullion. On the other hand, it is far from safe to assume that a ductile bar is homogeneous.

The alloys of gold and silver, and gold and copper form continuous series of solid solutions, and they may carry small amounts of other metals without showing marked segregation. In several of the examples cited below, remarkable agreement in the assays of different samples of gold and silver alloys containing small amounts of impurities shows the practical absence of segregation. Alloys of gold and copper are quite similar in their heat-behavior to the gold-silver alloys, but I do not happen to have any satisfactory series of samples of these carrying small amounts of other metals.

When we come to the ternary alloys our knowledge of their behavior on heating and cooling is too scanty to permit valid generalizations, and this is even more emphatically true of the alloys of increasing complexity.

Of general miscellaneous gold-bullion, there is only one universally satisfactory method of taking a sample that shall truly represent the metal sampled. This consists in pouring a small portion of the well-mixed molten metal into water, so as

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to produce small globules or granulations of the metal. As this is generally done by dipping out a portion of the molten metal in a small cup, such a sample is frequently called a "dip," and this name is generally used throughout the present paper. These samples are, however, often called "granulations;" and they are sometimes made by pouring directly out of the crucible into the water, the operation of casting being interrupted for the purpose.

As already indicated, there are indeed various cases where other styles of sampling may be sufficiently satisfactory, and there are, moreover, many cases where it is desirable or even necessary to sample a bar of solid bullion without melting it. In such cases a chip may be cut off from the bar with an ordinary cold-chisel, or a chisel specially designed for the purpose. Power-driven punches with special tools may be used. Machines are also built which bite out a triangular piece of metal by means of a projection on a lever operated by a cam. Where much chip-sampling is done, especially on small bars, these machines save much time and labor. A third method of sampling consists in boring into the bar, generally with a power-drill, and using the drillings for the assay-sample.

There is but little choice in the location of a chip-sample. It must necessarily be taken from a corner or along an edge of the bar. In the systematic sampling of large bars, generally two chips are cut—one from the top and one from the bottom of the bar—and properly identified.

In taking drill-samples there is a wide choice in the location of the drill-holes and in the sampling of large bars more or less of a plan in placing the holes is often followed. It is a common practice in the Mint Service to drill half-way through a bar at diagonally opposite corners of the top, and unite the drillings for the top-sample. The remaining corners are drilled half-way through from the bottom, and the drillings are mixed for the bottom-sample. Occasionally, especially when sampling very base bars, the four drillings are kept separate; and sometimes holes are drilled near the center of the bar also.

Drill-samples are often more satisfactory than chip-samples, especially where large numbers of bars of fairly uniform size and composition are sampled in accordance with a well-designed plan. Drill-samples of brittle bars are, however, liable

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to be unsatisfactory, because the fine and coarse portions may differ considerably in composition.

In the purchase of gold-bullion by the Mint Service of the United States, the size of the deposit has an important bearing upon the question of sampling. A very large proportion of the deposits will weigh less than 100 oz. each. Manifestly, slight differences in the samples on such bars will be immaterial. When, however, the weight of a deposit reaches 300 oz. the samples become important, and with bars weighing from 700 to 1,200 oz. correct sampling is essential.

Limiting myself for the most part to these large bars of miscellaneous bullion, I propose to illustrate by specific cases, drawn from actual practice, some of the general principles underlying the sampling of gold-bullion. In considering these illustrations, the assay-results must not be too rigidly interpreted. Besides the variations in the results due to variations in the samples, there are three other variants that must be kept in mind constantly: the chapter of accidents; the personal equation of the assayer; and the effect of the composition of the metal upon the assaying as distinct from its effect upon the sampling. This last point will be specially dealt with by various illustrations. Making reasonable allowances for these variants, it is thought that the illustrations are sufficiently conclusive upon the main points involved.

To begin with alloys of gold and silver containing only small amounts of base metal, four bars, each one carrying less than 4 base, were chipped top and bottom, and each one of the eight samples was assayed in three laboratories. A fifth bar, carrying 10 base, was treated in the same way. Table IV. summarizes the results reported.

TABLE IV.—*Sampling Gold-Silver Alloys Low in Base by Chips.*

Fineness.	Assays.	Fineness.	Assays.	Fineness.	Assays.	Fineness.	Assays.	Fineness.	Assays.
872.3	1 ...	875.7	... 1	883.3	1 1	884.2	2 ...	873.8	... 1
872.4	2 ...	875.8	1 2	883.4	... 6	884.3	... 1	874.0	1 ...
872.5	5 4	875.9	2 3	883.5	4 2	884.4	1 4	874.2	2 1
872.6	... 2	876.0	4 2	883.6	2 1	884.5	1 2	874.3	1 ...
872.7	... 3	876.1	1 1	883.7	3	884.6	1 ...	874.4	2 1
873.0	1 1	876.2	2 1	.....	.....	884.7	... 1	874.5	2 4
.....	.....	.....	.....	.....	.....	.....	.....	874.6	... 1
.....	.....	.....	.....	.....	.....	.....	.....	874.7	1 ...
Total...	9 10		10 10		10 10		5 8		9 8
Silver...	124		121		114		111		116

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Each one of the above bars may be taken as a single product, but a bar obtained by melting 39 small bars of miscellaneous origin into a mass-melt exhibits a remarkable agreement of chip-samples. This mass weighed a little more than 1,300 oz. and was 143 fine in silver. It was chipped, top and bottom, at the assay-office, and twice at the mint of re-deposit. The assay-office samples were assayed in duplicate there, and the four mint samples were assayed in four laboratories; 44 assays were made on the six chip-samples. Table V. summarizes the results reported:

TABLE V.—*Sampling a Mass-Melt by Chips.*

1 assay showed	679.7 fine in gold.
4 assays showed	679.8 fine in gold.
2 assays showed	679.9 fine in gold.
2 assays showed	680.1 fine in gold.
5 assays showed	680.2 fine in gold.
8 assays showed	680.3 fine in gold.
3 assays showed	680.4 fine in gold.
4 assays showed	680.5 fine in gold.
7 assays showed	680.6 fine in gold.
5 assays showed	680.7 fine in gold.
2 assays showed	680.8 fine in gold.
1 assay showed	680.9 fine in gold.
44	

A bar carrying more than 90 base and weighing 606 oz. was chipped, top and bottom, and the samples were assayed at the purchasing office. At the receiving-mint it was drilled, top and bottom, and the drillings were assayed in four service laboratories. Table VI. summarizes these assays.

TABLE VI.—*Chip- and Drill-Samples Agreeing.*

Fineness.	Chip.	Drill.
741.6	...	1
741.7	...	1
741.8	...	
741.9	...	2
742.0	...	2
742.1	2	3
742.2	2	1
742.3	...	4
742.4	...	5
Total	4	20
Silver	165	

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A bar weighing more than 800 oz. presents another remarkable case of the presence of considerable base metal without marked segregation. This bar was drilled, top and bottom, and the drillings were united to form one sample. Two other sets of drillings, top and bottom, were taken from the same bar, and each sample kept separate. These five samples were assayed 69 times in five laboratories. Table VII. summarizes the results reported.

TABLE VII.—*Sampling Gold-Silver Alloy, Containing Considerable Base, by Drilling.*

6 assays showed	351.0 fine in gold.
10 assays showed	351.1 fine in gold.
13 assays showed	351.2 fine in gold.
14 assays showed	351.3 fine in gold.
15 assays showed	351.4 fine in gold.
6 assays showed	351.5 fine in gold.
2 assays showed	351.6 fine in gold.
3 assays showed	351.7 fine in gold.
69	

This bar was 604 fine in silver and 44.5 fine in base metals.

Two other bars containing less base metal, but also considerably less gold, yielded much less satisfactory results. These bars weighed nearly 1,200 oz. each. They were drilled, top and bottom, and the drillings united to form one sample. One bar was drilled twice again, top and bottom, and the other three times, top and bottom, each drilling being kept separate. These 12 samples were assayed in five laboratories and Table VIII. summarizes the results.

On the other hand, a ductile bar 778 fine in gold with only 9 base yielded unsatisfactory results for this grade of metal. This bar weighed more than 1,500 oz. and was drilled, top and bottom, the drillings being mixed for one sample. Being ductile, it was chipped twice, top and bottom, at the mint of re-deposit. These six samples were kept separate. The bar was remelted with a loss of 0.38 oz. and two dip-samples were taken. The cold bar was again drilled, top and bottom. These 11 samples were assayed in four service laboratories, and Table IX. summarizes the results reported.

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TABLE VIII.—*Drill-Sampling Gold-Silver Alloy Low in Gold and Base.*

	1st Bar.				2d Bar.			
Fineness.	1st Drill.	2d Drill. T. B.	3d Drill. T. B.	Fineness.	1st Drill.	2d Drill. T. B.	3d Drill. T. B.	4th Drill. T. B.
290.2	.....	.....	2 ...	290.6	1	.....	.....	.....
290.6	1	1 ...	2 ...	290.9	1	.....	.....	.....
290.7	.....	1 ...	2 ...	291.1	1	1 ...	.....	.....
290.8	.....	1 ...	2 ...	291.5	1	.....	.....	.....
290.9	.....	2 ...	2 ...	291.9	1	2 ...	.....	.....
291.0	.....	.....	3 ...	292.0	1	2 ...	.....	1 ...
291.1	1	1 ...	1 ...	292.1	2	.....	.....	.....
291.2	.....	2 ...	.....	292.2	.....	2 ...	... 2	2 ...
291.3	.....	2 ...	1 ...	292.3	1	1 ...	... 1	3 1
291.4	2	2 1	... 2	292.4	.....	2 1	... 4	2 3
291.5	4	... 1	.....	292.5	3	2 2	... 2	3 3
291.6	2	1 ...	... 2	292.6	4	... 4	... 1	1 2
291.7	2	.....	.....	292.7	.....	1 4	... 1	1 3
291.8	.....	.....	... 1	292.8	.....	1 1	... 1	1 ...
291.9	1	... 1	... 1	292.9	.....	.....	.....	... 2
292.0	.....	.....	... 1	293.0	.....	... 2	1 1	.....
292.1	.....	... 1	.....	293.1	.....	.....	4 1	1 ...
292.2	2	... 2	.....	293.2	.....	1 ...	4 ...	.....
292.3	.....	.. 2	... 1	293.3	.....	.....	2 ...	.....
292.4	.....	1 3	.....	293.6	.....	.....	1 ...	.....
292.5	.....	... 1	.....	293.9	.....	.....	1 ...	.....
292.7	.....	... 1	... 1	.....	.....	.....	.....	.....
292.8	.....	... 1	... 2	.....	.....	.....	.....	.....
292.9	.....	.....	... 1	.....	.....	.....	.....	.....
293.0	.....	... 1	... 1	.....	.....	.....	.....	.....
293.1	.....	... ...	... 1	.....	.....	.....	.....	.....
Total...	15	14 15	15 14		16	15 14	13 14	15 14
Silver...		670					671	

TABLE IX.—*Sampling Ductile Gold-Silver Alloy, Low in Base.*

	1st Drill.	2d Drill. T. B.	1st Chip. T. B.	2d Chip. T. B.	Remelt. Drill. T. B.	Remelt. Dip. T. B.
Fineness.						
777.2	.....	.....	.....	1 ...	.....	.....
777.3	.....	.....	.....	2 ...	.....	.....
777.5	.....	.....	.....	.....	.....	.....
777.6	.....	.....	.....	.....	.....	.....
777.7	.....	.....	.....	3 ...	1 ...	.....
777.8	1	.....	.....	.....	.....	.....
777.9	.....	.....	... 1	.....	.....	.....
778.0	.....	.....	... 1		1 ...	.....
778.1	1	.....	.....	... 2	2 ...	.....
778.2	1	.....	2 2	... 1	2 ...	.....
778.3	1	3 2	2 3	... 3	2 2	... 1
778.4	1	1 4	... 1	... 3	2 ...	... 3
778.5	3	4 3	... 1	.....	2 3	2 5
778.6	3	1 2	... 2	... 1	... 5	6 1
778.7	1	.....	1 ...	.....	... 2	1 2
778.8	.....	.....	.....	.....	.....	1 ...
778.9	.....	1 ...	.....	.....	.....	1 ...
779.0	.....	.....	1 ...	.....	.....	.....
Total.....	12	10 11	6 11	6 10	12 12	11 12
Silver.....	212.5					

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The following case exhibits what is probably the very best agreement between various samples, and between an assay-office and a mint, that can be attained under every-day working-conditions:

A deposit of approximately 3,000 oz. was melted in the assay-office, and two dip-samples were taken. It was cast into three bars, and two chip-samples were taken from each bar. At the mint of re-deposit, two chip-samples were cut from each bar. The mint samples were forwarded to the Bureau, and thence sent to the assay-office and afterwards returned to the mint to be assayed, neither institution being informed what the samples were.

In all, ten samples were taken from this deposit, and 47 assays were made at the two institutions. Table X. summarizes the assays reported.

TABLE X.—*Agreement of Assays on Various Samples at Two Institutions.*

1 assay showed	875.5 fine in gold.
1 assay showed	875.6 fine in gold.
4 assays showed	875.7 fine in gold.
4 assays showed	875.8 fine in gold.
14 assays showed	875.9 fine in gold.
12 assays showed	876.0 fine in gold.
6 assays showed	876.1 fine in gold.
4 assays showed	876.2 fine in gold.
1 assay showed	876.3 fine in gold.
47	

If we eliminate the results that were reported only once each, we have 44 assays, ranging from 875.7 to 876.2. This deposit might, therefore, be properly reported by either institution at either 875.75 or 876. It did not contain over 4 base.

On three bars the purchasing office took chip- and dip-samples, and on a fourth bar took dip- and drill-samples. The mint of re-deposit cut two chip-samples from each bar. These samples were assayed in various service laboratories, and Table XI. summarizes the results on the various styles of samples.

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TABLE XI.—*Sampling in Various Ways with Satisfactory Agreement.*

Fineness.	Dip.	Chip.	Chip.	Fineness.	Dip.	Drill.	Chip.	Fineness.	Dip.	Chip.	Chip.	Fineness.	Dip.	Chip.	Chip.
861.4	...	1	2	895.1	1	...	2	858.5	1	...	1	883.2	...	...	1
861.5	...	...	1	895.2	...	...	3	858.7	...	1	...	883.3	...	2	...
861.7	1	1	...	895.3	...	1	4	858.9	...	...	1	883.4	1	1	1
861.8	...	4	1	895.4	1	...	2	859.0	1	3	1	883.5	2	1	6
861.9	3	1	4	895.5	1	2	1	859.1	1	1	2	883.6	...	...	1
862.0	3	3	6	895.6	1	1	2	859.2	3	3	7	883.7	...	1	4
862.1	3	1	3	895.7	2	2	1	859.3	1	1	2	883.8	5	4	...
862.2	1	1	...	895.8	3	3	...	859.4	3	...	1	883.9	3	3	3
862.3	2	1	...	895.9	3	4	...	859.5	5	1	...	884.0	3	2	1
862.4	1	...	...	896.0	1	...	1	859.6	...	3	3	884.1	...	...	1
.....	...	...	...	896.1	1	...	...	859.7	1	...	...	884.2	...	...	1
.....	...	...	...	896.2	...	1	...	.....	...	...	...	.....	...	...	...
Totals...	14	13	17		14	14	16		16	13	18		14	14	19
Silver...	134					98			137				113		

On three bars the purchasing office took dip- and drill-samples, and the mint of re-deposit took top and bottom drill-samples. These samples were assayed in various service laboratories, and the results are summarized in Table XII., showing a considerable agreement between the dip-samples and the drills taken at the separate offices. But a better agreement is desirable.

TABLE XII.—*Dip- and Drill-Samples Partly Satisfactory.*

Fineness.	Dip.	Drill.	Drill.	Fineness.	Dip.	Drill.	Drill.	Fineness.	Dip.	Drill.	Drill.
822.4	1	.....	.....	973.4	.....	1	.....	972.1	.....	1	.....
822.7	1	.....	.....	973.5	.....	1	.....	972.2	1	1	.....
822.8	.....	.....	2	973.6	.....	.....	1	972.3	3	.....	3
822.9	1	1	1	973.8	.....	1	1	972.4	4	1	.....
823.0	3	1	2	974.0	.....	.....	2	972.5	.....	.....	1
823.1	3	2	1	974.1	.....	1	.....	972.6	3	1	1
823.2	2	4	.....	974.2	2	.....	2	972.7	1	.....	2
823.3	.....	3	1	974.3	2	.....	.....	972.8	.....	.....	2
823.4	1	2	4	974.4	1	.....	1	972.9	.....	1	.....
823.5	1	.....	.....	974.5	2	1	2	973.0	.....	2	2
823.6	.....	.....	2	974.6	1	.....	.....	973.1	.....	3	.....
823.7	.....	1	1	974.7	.....	1	2	973.2	1	.....	.....
823.8	.....	.....	4	974.8	3	2	4	973.3	1	1	1
823.9	.....	.....	1	974.9	1	.....	2	973.4	.....	1	1
824.1	.....	.....	1	975.0	... ..	4	1	973.5	.....	.....	2
.....	.....	.....	.....	975.1	1	1	.....	973.6	.....	.....	1
.. ...	.....	.....	.....	975.2	.....	1	2	973.8	.....	1	1
.....	.....	.....	.....	975.3	1	.....	1	974.3	.....	1	.....
. ....	.....	.....	.....	.....	.....	.....	.....	975.7	.....	.....	2
Totals...	13	14	20		14	14	21		14	14	19
Silver...		162				4			Practically none.		

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On four bars the purchasing office took dip- and drill-samples and the mint of re-deposit cut two chips from each bar. These samples were assayed in various laboratories in the service, and Table XIII. summarizes the results reported, showing the chip-samples to be unsatisfactory.

TABLE XIII.—*Chip-Samples Unsatisfactory.*

Fineness.	Dip.	Drill.	Chip.	Fineness.	Dip.	Drill.	Chip.	Fineness.	Dip	Drill.	Chip.	Fineness.	Dip.	Drill.	Chip.
821.8	...	...	1	892.3	...	...	1	898.0	...	...	1	820.7	1	...	
822.3	...	...	3	892.6	...	...	2	898.1	...	...	4	820.8	...	...	2
822.5	...	...	1	892.7	...	...	5	898.2	...	...	2	820.9	...	...	2
822.6	...	...	1	892.8	...	...	2	898.3	...	1	3	821.0	4	...	1
822.7	2	1	1	892.9	1	...	4	898.4	1	2	2	821.1	1	...	...
822.9	...	...	1	893.1	...	2	...	898.6	1	2	1	821.2	2	...	...
823.0	1	1	2	893.2	1	1	1	898.7	3	...	1	821.3	1	...	...
823.1	4	3	3	893.3	4	2	1	898.8	2	2	1	821.4	3	2	...
823.2	3	...	...	893.4	...	1	...	898.9	3	3	...	821.5	1	1	...
823.3	1	2	...	893.5	5	4	...	899.0	3	2	...	821.6	...	1	...
823.4	...	1	1	893.6	3	...	...	899.1	1	...	1	821.7	1	3	1
823.5	4	2	1	893.7	...	1	1	899.4	...	...	1	821.8	1	3	1
823.6	1	3	2	893.8	...	1	...	899.8	...	1	...	821.9	...	...	2
.....	...	...	...	893.9	...	1	...	899.9	...	1	...	822.0	...	1	2
.....	...	...	...	.....	...	...	...	.....	...	...	...	822.1	...	2	...
.....	...	...	...	.....	...	...	...	.....	...	...	...	822.5	...	...	1
Totals..	16	13	17		14	13	17		14	14	17		15	13	12
Silver...		160				97				88				172	

On three bars the purchasing office took dip- and chip-samples and the receiving mint cut two chips. These samples were assayed in various laboratories in the service, and Table XIV. summarizes the results reported, showing that the mint chip-samples were entirely unsatisfactory.

A very complete set of samples from five bars, contained in one shipment made by a purchasing office to a mint for re-deposit, shows what may happen under every-day working-conditions in handling bullion of moderate grade. Everything considered, the first set of samples may be taken as fairly satisfactory, but the last set is totally unsatisfactory. At the office of purchase two dips and two drills were taken from each bar. At the mint of re-deposit two drills and two chips were taken. Each one of these eight samples on each bar was assayed in various service laboratories, and Table XV. summarizes the results reported; the assays on each set of two samples being grouped.

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TABLE XIV.—*Second Chip-Samples Unsatisfactory.*

Fineness.	Dip.	Chip.	Chip.	Fineness.	Dip.	Chip.	Chip.	Fineness.	Dip.	Chip.	Chip.
879.8	...	...	1	890.3	...	...	1	867.6	...	...	1
880.0	...	...	1	890.5	...	...	2	867.8	...	...	1
880.1	...	...	2	890.6	...	...	2	868.1	...	...	2
880.6	...	...	1	890.8	...	...	1	868.2	...	...	1
881.3	...	...	1	890.9	...	...	4	868.3	...	...	1
881.4	...	...	1	891.0	...	...	1	868.5	...	...	4
881.5	...	...	2	891.1	...	...	1	868.6	...	...	4
881.6	...	...	1	891.2	...	...	1	868.7	...	...	1
881.7	...	...	2	891.3	...	2	1	868.8	...	1	...
881.9	...	1	...	891.4	2	...	...	869.0	...	1	1
882.0	...	1	1	891.5	...	2	...	869.3	1	...	...
882.1	...	...	1	891.6	4	1	...	869.5	...	1	...
882.2	...	2	...	891.7	5	1	...	869.6	1	5	...
882.3	2	3	...	891.8	2	4	...	869.7	1	1	...
882.4	3	1	...	891.9	...	5	...	869.8	2	1	...
882.5	2	4	...	892.0	4	1	...	869.9	1	1	...
882.6	2	...	1	892.2	1	3	...	870.0	1	2	...
882.7	4	...	...	892.3	...	1	...	870.1	3	1	...
882.8	...	3	...	.....	...	...	...	870.2	1	2	...
882.9	2	...	...	.....	...	...	...	870.3	4	2	...
883.0	3	1	...	.....	...	...	...	870.4	1	...	...
883.1	1	1	...	.....	...	...	...	870.5	2	1	...
883.2	...	1	...	.....	...	...	...	870.6	1	...	...
.....	...	...	...	.....	...	...	...	870.7	1	...	...
.....	...	...	...	.....	...	...	...	870.9	1	...	...
.....	...	...	...	.....	...	...	...	871.0	1	...	...
.....	...	...	...	.....	...	...	...	871.1	...	2	...
Totals...	19	18	15		18	20	14		22	21	16
Silver...		90				98				95	

TABLE XV.—*Sampling Five Bars of Miscellaneous Bullion.*

Fineness.	Dip.	Drill.	Drill.	Chip.	Fineness.	Dip.	Drill.	Drill.	Chip.	Fineness.	Dip.	Drill.	Drill.	Chip.
630.3	...	1	1	...	755.0	...	...	1	1	544.2	...	...	...	1
630.4	...	1	...	1	755.1	...	...	1	...	544.3	...	...	...	1
630.5	...	...	...	2	755.2	...	...	...	2	544.6	...	...	...	1
630.6	3	...	..	...	755.3	...	...	1	2	544.7	...	...	...	1
630.7	1	1	2	3	755.4	...	...	3	2	544.9	...	...	...	1
630.8	...	...	2	1	755.5	...	1	1	...	545.0	...	...	...	2
630.9	2	1	2	1	755.6	...	...	1	4	545.1	...	...	1	1
631.0	2	2	2	2	755.7	...	...	...	1	545.2	...	...	...	1
631.1	3	...	...	...	755.9	...	1	1	...	545.3	...	...	...	2
631.2	2	2	...	1	756.0	...	...	1	...	545.4	1	...	...	1
631.3	1	2	1	...	756.1	...	1	1	1	545.5	..	...	1	...
631.4	1	2	...	...	756.2	...	2	3	1	545.6	...	...	1	1
631.5	1	2	1	1	756.3	...	1	...	...	545.7	...	...	1	...
631.6	...	...	2	...	756.4	...	1	...	...	545.8	...	1	1	1
631.7	...	1	1	...	756.5	1	...	1	...	545.9	2	...	1	...
.....	...	...	...	...	756.6	2	...	1	...	546.0	2	...	...	...
.....	...	...	...	...	756.7	2	2	...	...	546.1	3	...	1	...
.....	...	...	...	...	756.8	1	1	...	...	546.2	1	...	1	...
.....	...	...	...	...	756.9	1	1	...	...	546.3	1	2	...	...
.....	...	...	...	...	757.2	3	...	...	...	546.4	3	1	1	...
.....	...	...	...	...	757.3	1	1	...	...	546.5	2	2	...	...
.....	...	...	...	...	757.4	2	...	...	...	546.6	1	2	1	...
.....	..	...	...	...	757.5	1	...	...	...	546.7	...	1	...	...
.....	...	...	...	...	.....	...	...	...	...	546.8	...	1	1	...
.....	...	...	...	...	.....	...	...	...	...	546.9	2	1	...	...
.....	...	...	...	...	.....	...	...	...	...	547.0	...	...	1	...
.....	...	...	...	...	.....	...	...	...	...	547.2	...	2	1	...
.....	...	...	...	...	.....	...	...	...	...	547.3	...	...	1	...
Totals...	16	15	14	12		14	12	16	14		18	13	14	14
Silver...			330					203					444	

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Fineness.	Dip.	Drill.	Drill.	Chip.	Fineness.	Dip.	Drill.	Drill.	Chip.
549.4	...	...	...	1	568.5	...	...	...	1
549.7	...	...	...	1	569.6	...	...	...	1
550.5	...	...	...	1	573.9	...	...	...	1
551.0	...	...	...	1	574.6	...	...	...	1
551.4	...	...	...	1	575.4	...	...	...	1
551.7	...	...	...	2	575.7	...	...	...	1
551.8	...	...	...	1	576.4	...	...	...	1
551.9	1	...	1	...	576.5	...	...	...	1
552.1	1	...	...	2	577.0	...	...	...	2
552.2	...	...	1	...	577.2	...	...	...	1
552.3	...	...	...	1	577.3	...	...	1	...
552.5	...	...	1	1	577.4	...	...	...	1
552.6	1	...	3	...	577.6	...	...	1	...
552.7	3	...	2	1	577.8	...	...	1	...
552.8	2	1	1	...	577.9	...	...	1	...
552.9	4	4	3	...	578.0	...	...	2	...
553.0	2	...	...	...	578.1	...	...	1	...
553.1	4	5	1	...	578.3	1	...	...	...
553.2	3	3	...	...	578.4	...	1	...	...
553.3	...	3	...	...	578.5	...	1	1	...
553.4	...	1	1	...	578.6	...	3	2	...
553.5	1	...	...	...	578.7	3	2	2	1
553.6	...	...	2	...	578.8	3	1	...	...
.....	...	...	...	...	578.9	4	2	1	...
.....	...	...	...	...	579.0	4	1	1	...
.....	...	...	..	...	579.1	...	3	...	...
.....	...	...	...	...	579.2	1	1	...	1
.....	...	...	...	...	579.3	2	..	...	...
.....	...	...	...	...	579.4	1	1	...	...
.....	...	...	...	...	579.5	1	...	...	...
.....	...	...	...	...	579.6	...	1	...	...
.....	...	...	...	...	579.7	...	1	...	...
	—	—	—	—		—	—	—	—
Totals...	22	17	16	13		20	18	14	14
Silver....		410						335	

A small bar made by melting up scrap-material has a most interesting and instructive sample-history. The bar weighed a trifle more than 31 oz. and was shipped from the assay-office as 826 fine in gold. At the receiving mint it was drilled, top and bottom, and while the assays on these samples agreed fairly well, yet they averaged considerably less than 826. The bar was again drilled, top and bottom, and these samples yielded very erratic assays. The bar was then melted and two dip-samples were taken. The solid bar was again drilled, top and bottom. These eight samples were assayed in four service laboratories, and Table XVI. summarizes the results reported.

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TABLE XVI.—*Sampling a Small Unsatisfactory Bar.*

Fineness.	Drill. T.	Drill. B.	Drill. T.	Drill. B.	Remelt. Drill. T.	Remelt. Drill. B.	Remelt. Dip. 1st.	Remelt. Dip. 2d.
823.0	.....	.....	.....	1	.....	.....	.....	.....
823.5	.....	.....	.....	1	.....	.....	.....	.....
823.7	.....	.....	.....	1	.....	.....	.....	.....
823.9	.....	.....	.....	1	.....	.....	.....	.....
824.3	.....	.....	.....	1	.....	.....	.....	.....
824.4	.....	.....	.....	2	.....	.....	.....	.....
824.7	.....	1	.....	.....	.....	.....	.....	.....
825.1	.....	.....	.....	.....	.....	.....	1	2
825.2	.....	2	1	.....	.....	.....	.....	.....
825.3	.....	1	1	.....	.....	.....	1	.....
825.4	2	2	.....	.....	.....	.....	1	.....
825.5	4	2	.....	.....	.....	.....	.....	.....
825.6	2	2	.....	.....	1	.....	3	1
825.7	1	.....	.....	.....	.....	1	.....	1
825.8	1	.....	1	.....	.....	3	3	.....
825.9	.....	.....	1	.....	.....	1	1	1
826.0	1	1	1	.....	2	.....	3	.....
826.1	.....	.....	.....	.....	1	1	.....	.....
826.2	.....	.....	.....	.....	2	2	.....	2
826.3	.....	.....	2	.....	.....	1	.....	2
826.4	.....	.....	.....	.....	1	.....	.....	1
826.5	.....	.....	1	.....	1	.....	.....	2
826.6	.....	.....	2	.....	2	.....	.....	.....
827.0	.....	.....	1	.....	.....	.....	.....	.....
Totals.....	11	11	11	7	10	9	13	12
Silver.....		143						

On three bars drills were taken from diagonally opposite corners on top of the bar and also near the center of the bar. The other two corners were drilled from the bottom of the bar and a drilling was made near the center of the bottom. All the top-drillings were mixed to make one sample, and all the bottom-drillings to make another. These samples were assayed in two laboratories, and Table XVII. summarizes the results reported, to which are added for comparison duplicate assays of two dip-samples of each bar made in one of the laboratories.

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TABLE XVII.—*Sampling by Top and Bottom Drills.*

Fineness.	1st Bar. Bottom.	Top.	Fineness.	2d Bar. Bottom.	Top.	Fineness.	3d Bar. Bottom.	Top.
883.7	2	.....	884.8	3	.....	888.7	1	.....
883.9	4	.....	884.9	3	.....	888.8	1	.....
884.0	1	.....	885.0	4	.....	888.9	1	.....
884.1	6	.....	885.1	3	.....	889.0	2	.....
884.2	3	.....	885.2	3	.....	889.1	3	.....
884.3	1	.....	885.3	1	.....	889.2	2	.....
884.4	1	.....	885.4	5	.....	889.3	1	.....
884.5	1	.....	885.5	2	.....	889.4	2	.....
884.7	.....	1	885.6	1	.....	889.5	2	1
885.1	.....	5	885.7	1	.....	889.6	1	.....
885.2	.....	2	886.2	.....	4	889.7	1	.....
885.3	.....	1	886.3	.....	3	889.8	3	.....
885.4	.....	3	886.5	.....	6	890.0	.....	2
885.5	.....	2	886.6	.....	4	890.1	.....	3
885.6	.....	4	886.7	.....	1	890.2	.....	2
885.7	.....	2	886.8	.....	1	890.3	.....	1
.....	.....	.....	887.0	.....	1	890.6	.....	2
.....	.....	.....	.....	.....	.....	891.1	.....	1
.....	.....	.....	.....	.....	.....	891.3	.....	1
.....	.....	.....	.....	.....	.....	891.4	.....	1
.....	.....	.....	.....	.....	.....	891.6	.....	1
.....	.....	.....	.....	.....	.....	891.7	.....	1
.....	.....	.....	.....	.....	.....	891.8	.....	1
.....	.....	.....	.....	.....	.....	891.9	.....	1
.....	.....	.....	.....	.....	.....	892.2	.....	2
	19	20		26	20		20	20

Dip-Samples.		Dip-Samples.		Dip-Samples.	
884.0	3	885.3	1	889.1	2
884.1	1	885.4	1	889.2	2
		885.5	1		
		885.7	1		

With the exception of the single assay at 889.5 on the third bar, all the top-assays are above the highest bottom-assay on each bar. All the top-assays are above the dip-assays in each one of these cases. Each one of the bars was less than 5 fine in silver.

As a general proposition, a drill-sample will often be better and more satisfactory than a chip-sample, simply because it represents a larger volume of the metal; but a drill-sample of a brittle bullion may be open to a serious objection, because the fine particles of the sample are generally liable to be quite different in composition from the coarse portions, so that, even if the drill-sample as a whole should be fairly representative

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of the bar, yet it would be necessary to have the 0.5 g. weighed out for the assay composed of fine and coarse material in just the same proportions as the whole sample. Manifestly this will seldom be the case.

Drill-samples of two closely-related bars were sifted on an 80-mesh screen, and yielded the following results:

	1st Bullion.	2d Bullion.
Coarse portion.....	290.3 fine.	291.4 fine.
	290.4 fine.	291.5 fine.
Fine portion.....	287.8 fine.	290.1 fine.

Two other drills of these same bars were sifted. The portion between 20- and 40-mesh and that finer than 60-mesh were assayed, with the following results:

	1st Bullion.	2d Bullion.
20–40-mesh	290.4 fine.	291.9 fine.
	291.9 fine.	292.3 fine.
60-mesh	289.6 fine.	290.3 fine.
	290.3 fine.	290.8 fine.

Two other bullions treated like the last illustration showed:

	1st Bullion.	2d Bullion.
20–40-mesh.....	351.7 fine.	573.2 fine.
	351.9 fine.	573.4 fine.
60-mesh.....	350.2 fine.	566.6 fine.
	350.5 fine.	567.3 fine.

An entirely different bullion sifted differently showed:

40–60-mesh.....	585.3 fine.	100-mesh.....	584.6 fine.
	585.4 fine.		584.7 fine.

A very rich bullion sifted like the last showed:

40–60-mesh.....	889.2 fine.	100-mesh.....	887.8 fine.
	889.2 fine.		

Five samples of a bullion which had given discordant assays were subjected to sifting-tests, and the resulting samples were assayed. Four drill-samples were sifted on 100-mesh, and the coarse and fine portions yielded the following results:

A.		B.		C.		D.	
Coarse.	Fine.	Coarse.	Fine.	Coarse.	Fine.	Coarse.	Fine.
615	612.7	615.3	612.2	613.9	612.9	612.8	608.
617	614.4	615.9	612.3	615.1	613.1	616.6	609.4
		617.9	613.7		614.1		610.5

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Slightly more than a gram of the small pieces of a dip-sample was crushed and assayed in duplicate. Coarser pieces were crushed and sifted on 80-mesh. The three samples yielded the following results:

Whole Dip-Sample.	Crushed Dip-Sample.	
	Coarse.	Fine.
614.8	614.8	614.8
615.	615.2	614.9

This bullion carried bismuth.

No class of bullion has given assayers the world over so much trouble as the so-called cyanide-bullion. Not all bullion produced by cyanide-mills is troublesome. Hundreds of bars produced by such mills have passed through our Mint Service without the slightest trouble. It is easy enough to produce a high-grade refined bar from the zinc-box precipitate; but when this precipitate is melted direct and put into bars without proper refining, there is liable to be no end of trouble with the assays. Unfortunately, too, in some instances the precipitate is not as thoroughly cleaned from zinc before drying as it might be. These dirty unrefined bars are the ones that are usually spoken of as "cyanide-bars" with so much disparagement.

More than 15 years ago Roberts-Austen[2] gave a startling illustration of the difficulties and uncertainties of assaying this class of bullion. A bar weighing 393 oz. was sampled and assayed in the usual manner, and paid for at £965. The gold from this bar was separated and refined by itself, and was found to be worth £1,028. This meant a loss of £63, or more than $300, to the former owner of the bar.

It is generally assumed that the zinc remaining in the bullion is the cause of the trouble, but no clear and systematic explanation of its action has yet been given. It is quite probable that its action is complex, and differs in different bullions according to the presence or absence of other metals. I have, for instance, published[3] 50 assays made in eight laboratories on a synthetic bullion approximately 590 fine in gold, 245 in silver, 130 in zinc, with a little copper and a very little lead, which

[2] *Annual Report Deputy Master and Comptroller of the Mint*, No. 27, p. 38 (1896).

[3] *Trans.*, xl., 794 (1909). *Annual Report of the Director of the Mint for the Year* 1908–09, p. 34.

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ranged from 588.9 to 589.9 fine in gold. Clearly the zinc did not seriously interfere with the actual assaying in this case.

I have also published[4] 207 assays made on three bars of bullion produced in the celebrated Mercur mill, showing widely-varying assays. By a qualitative analysis, this bullion was found to carry both cadmium and nickel, and four other bullions from cyanide-mills in Montana showed these metals. At present I am carrying on, as occasion permits, a series of test-assays on synthetic alloys of gold, zinc, and cadmium.

From our tests in sampling cyanide-bullion I am satisfied that much of the variation ordinarily shown by the assays of this class of bullion is due to differences in the samples, arising from the effect of the zinc upon the physical structure of the metal; but, aside from this, there are many cases where the composition of the metal directly affects the assay-work itself.

TABLE XVIII.—*Effects of Composition Upon the Assay-Work.*

Fineness.	1st Dip.	2d Dip.	Fineness.	1st Dip.	2d Dip.	Fineness.	1st Dip.	2d Dip.
784.9	.....	1	816.6	.....	1	821.9	1	.....
785.2	.....	3	816.8	.....	2	822.0	2	.....
785.3	1	.....	817.0	.....	1	822.2	3	.....
785.4	3	1	817.2	.....	1	822.3	2	2
785.5	2	1	817.3	1	.....	822.4	1	3
785.6	1	2	817.4	2	1	822.5	1	2
785.7	3	.....	817.5	1	.....	822.6	2	.....
785.8	1	.....	817.6	3	2	822.7	2	.....
785.9	2	3	817.7	2	.....	822.8	4	1
786.0	1	1	817.8	1	1	822.9	1	.....
786.2	1	.....	817.9	2	3	823.0	.....	2
786.3	.....	1	818.0	1	1	823.1	.....	3
786.5	1	.....	818.1	3	.....	823.2	1	2
786.6	1	2	818.2	.....	2	823.5	.....	1
786.8	.....	1	818.3	1	1	823.6	.....	2
787.0	1	3	818.4	1	.....	823.7	.....	1
.....	.....	.....	818.5	.....	2	.....	.....	.....
.....	.....	.....	818.6	.....	2	.....	.....	.....
.....	.....	.....	818.7	.....	1	.....	.....	.....
.....	.....	.....	818.8	1	1	.....	.....	.....
.....	.....	.....	819.0	1	1	.....	.....	.....
.....	.....	.....	819.1	.....	1	.....	.....	.....
.....	.....	.....	819.2	2	.....	.....	.....	.....
.....	.....	.....	819.3	.....	1	.....	.....	.....
Totals.....	18	19		22	25		20	19

These bars ranged from 5 to 10 fine in silver.

[4] *Engineering and Mining Journal*, vol. xciii., No. 15, p. 733 (Apr. 13, 1912).

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The illustrations of top- and bottom-drilling given in Table XVII. are from cyanide-bars. Five of these samples show that individual samples of cyanide-bullion may give fairly satisfactory results, but that two such samples from the same bar may leave the question of the actual content of gold in the bar in doubt.

TABLE XIX.—*Sampling Cyanide-Bullion.*

1st Bar.			2d Bar.			3d Bar.		
Fineness.	Drills.	Dips.	Fineness.	Drills.	Dips.	Fineness.	Drills.	Dips.
826.9	1	.....	816.9	1	.....	831.5	1	.....
827.2	2	.....	817.2	1	.....	832.1	.....	1
827.3	1	.....	817.3	3	.....	832.3	.....	3
827.4	1	.....	817.5	1	.... .	832.4	.....	1
827.6	1	.....	818.3	.....	1	832.6	.....	1
827.7	3	.....	818.4	1	.....	832.8	1	2
827.8	1	.....	818.5	.....	1	832.9	1	1
828.3	.....	1	818.6	1	1	833.0	.....	4
828.4	2	.....	818.7	.....	1	833.1	1	3
828.5	4	.....	818.8	.....	1	833.3	1	2
828.6	1	1	818.9	1	2	833.4	1	3
828.7	1	4	819.0	1	1	833.5	3	6
828.8	2	1	819.1	2	1	833.6	1	1
829.0	1	3	819.2	.....	1	833.7	1	1
829.1	2	2	819.3	.....	1	833.8	2	1
829.2	3	3	819.4	.....	1	833.9	.....	1
829.3	1	5	819.5	1	3	834.0	1	2
829.4	1	2	819.6	2	3	834.1	1	2
829.5	1	5	819.7	.....	2	834.4	1	1
829.6	1	3	819.8	.....	2	834.7	1	.....
829.7	.....	1	819.9	.....	4	835.0	1	.....
829.8	.....	3	820.0	4	4	835.4	2	.....
829.9	2	1	820.1	.....	1	835.5	1	.... .
830.0	.....	4	820.2	2	2	835.6	1	.....
830.1	.....	1	820.3	.....	2	835.9	1	.....
830.3	.....	1	820.5	.....	1	.....	.....	.....
830.4	1	.....	820.6	1	2	.....	.....	.....
830.5	.....	1	820.7	2	1	.....	.....	.....
830.8	1	.....	820.8	2	.....	.....	.....	.....
.....	.....	.....	820.9	2	.....	.....	.....	.....
.....	.....	.....	821.0	1	1	.....	.....	.....
.....	.....	.....	821.2	1	.....	.....	.....	.....
.....	.....	.....	821.4	1	.....	.....	.....	.....
.....	.....	.....	821.5	1	.....	.....	.....	.....
.....	.....	.....	821.8	1	.....	.....	.....	.....
.....	.....	.....	822.1	1	.....	.....	.....	.....
	—	—		—	—		—	—
Totals.....	34	42		34	40		23	36

All of these bars carried less than 5 of silver.

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Again, three bars from the same mill were sampled by dips, twice each, and the six samples were assayed in various laboratories in the service, giving widely varying results, and showing the effect of the composition of the metal upon the actual assaying. Table XVIII. summarizes the results reported.

Three bars deposited at one time by a cyanide-mill furnish an excellent illustration of the general conditions in handling this class of bullion. Two drills and two dips were taken from each bar, and the 12 samples were sent to various laboratories in the service for assay. Table XIX. summarizes the results reported, together with the original assays at the office of deposit, the assays on each kind of sample being grouped.

In one office receiving a great deal of high-grade bullion it is the general experience that the chip-samples will run somewhat below the dip-samples on this rich material. Illustrations already given show that no fixed relation is generally exhibited between the dip- and drill-samples as to the tenor of gold. Where many bars of practically uniform composition are drilled in accordance with a well-designed plan, a more or less fixed relationship between the drill- and dip-assays might appear. Such a plan was developed at one time for certain deposits at one of our mints.

Table XX. shows that with cyanide-bars a drill-sample may give much higher assays than a dip-sample. On the first bar, 13 assays of the drill-samples are above the highest assay on the dip-samples; and on the second bar, 22 drill-assays exceeded the highest dip-assays. Two cyanide-bars, each weighing about 850 oz. and practically free from silver, were drilled, top and bottom, and these four samples, together with four corresponding dips, were sent to various service laboratories for assay. Table XX. summarizes the results reported, together with the original mint-assays, the assays on each kind of sample being grouped.

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TABLE XX.—*Sampling Cyanide-Bullion by Drills and Dips.*

1st Bar. Fineness.	Drills.	Dips.	2d Bar. Fineness.	Drills.	Dips.
894.7	1	.....	905.6	1	.....
894.8	2	.....	906.1	2	.....
895.0	1	.....	906.2	2	.....
895.2	1	.....	906.6	.....	1
895.3	.....	1	906.8	1	1
895.5	1	.....	906.9	1	.....
895.7	.....	1	907.0	1	.....
895.8	2	2	907.1	1	1
895.9	1	.....	907.2	.....	2
896.0	1	.....	907.3	.....	2
896.1	1	1	907.4	.....	2
896.2	2	1	907.5	1	2
896.3	3	3	907.6	.....	2
896.4	2	1	907.7	2	4
896.5	2	3	907.8	1	2
896.6	3	1	907.9	2	5
896.7	2	3	908.0	2	.....
896.8	1	2	908.1	1	.....
896.9	1	2	908.3	1	1
897.0	1	.....	908.4	2	.....
897.1	.....	1	908.5	2	.....
897.2	.....	1	908.7	1	1
897.5	1	.....	908.9	1	.....
898.0	1	.....	909.0	1	.....
898.1	2	.....	909.4	1	.....
898.6	1	.....	909.5	3	.....
898.7	1	.....	909.6	2	.....
899.2	1	.....	909.7	1	.....
899.4	1	.....	909.8	1	.....
899.6	2	.....	909.9	1	.....
899.8	3	.....	910.3	2	.....
.....	.....	.....	910.4	1	.....
.....	.....	.....	910.5	1	.....
.....	.....	.....	910.7	1	.....
.....	.....	.....	910.8	1	.....
.....	.....	.....	911.0	1	.....
.....	.....	.....	911.4	1	.....
.....	.....	.....	911.8	1	.....
.....	.....	.....	911.9	1	.....
.....	.....	.....	912.0	1	.....
	41	23		46	26

Undoubtedly much of the trouble with these cyanide-bars would be avoided by a better cleaning of the zinc-box precipitate. It is, however, easy enough to refine the impure bullion by strongly-oxidizing fusion in the crucible; but this is expen-

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sive in labor and reagents, and there is some loss of gold. I have previously published[5] an illustration of this.

A cyanide-bar weighing 643.30 oz. was melted, with a loss of 6.07 oz., and 10 assays on various kinds of samples were made, with the following results:

1 assay showed 844.6 fine in gold.
1 assay showed 846.3 fine in gold.
1 assay showed 846.6 fine in gold.
2 assays showed 847.0 fine in gold.
1 assay showed 847.2 fine in gold.
2 assays showed 847.6 fine in gold.
1 assay showed 847.8 fine in gold.
1 assay showed 848.0 fine in gold.

10

The bar was melted seven times, when it weighed 502.01 ounces, showing a total loss of 141.29 and an estimated loss of 3.75 oz. of gold. Eight assays were made on various samples of the final metal, with the following results:

1 assay showed 933.2 fine in gold.
2 assays showed 933.3 fine in gold.
2 assays showed 933.4 fine in gold.
1 assay showed 933.5 fine in gold.
2 assays showed 933.7 fine in gold.

8

The final bar was 21 fine in silver.

A great deal of old scrap material of the most heterogeneous character is purchased at some of our offices, and many of these deposits are small. On these small bars the question of sampling and assaying is not so important; but before the purchasing offices ship these small deposits to a mint they are united into mass-melts, when difficulties may appear in the sampling, and become important. Nine small bars were united to make a mass-melt of 338 oz. Dip- and drill-samples were taken at the purchasing office, and the bar was drilled, top and bottom, at the receiving mint. These samples were assayed in various service laboratories, and Table XXI. summarizes the results reported.

[5] *Trans.*, xl., 789 (1909). *Annual Report of the Director of the Mint for the Year* 1908–09, p, 31.

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TABLE XXI.—*Sampling Mass-Melt.*

Fineness.	Dip.	Drill.	Drill.
500.7	.....	.....	1
500.8	.....	.....	1
501.1	.....	.....	1
501.7	.....	.....	1
501.8	.....	.....	2
501.9	1	.....	.....
502.0	.....		.....
502.1	.....	1	.....
502.4	4	.....	2
502.5	.....	.....	2
502.6	.....	.....	2
502.8	1	.....	1
502.9	1	1	1
503.0	1	1	1
503.1	.....	2	1
503.2	.....	1	.....
503.4	.....	2	1
503.5	1	.....	.....
503.6	.....	2	.....
503.7	1	.....	.....
503.8	.....	.....	1
503.9	2	1	.....
504.0	.....	.....	1
504.2	.....	.....	2
504.7	1	1	.....
Totals.....	13	14	21
Silver.....		225	

What is probably the very worst case of disagreeing samples and assays ever investigated by the Mint Bureau was a bar weighing 774.39 oz., carrying nearly 400 base, largely copper, which was shipped to a mint as being 568 fine in gold. Three sets of drills, top and bottom, were taken, and, these proving very unsatisfactory, the bar was remelted, with a loss of 4.6 oz. Two dip-samples were taken, and the bar was again drilled, top and bottom. These ten samples were assayed in four laboratories in the service and 113 assays were made. Table XXII. summarizes these assays.

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TABLE XXII.—*Sampling Bar* 400 *Base, Largely Copper.*

Original Bar.							Remelt.			
	1st Drill.		2d Drill.		3d Drill.			Drill.		Dip.
Fineness.	T.	B.	T.	B.	T.	B.	Fineness.	T.	B.	A.	B.
564.7	.....	.....	.....	.....	.....	1	571.2	.....	2	.....	.....
565.3	.....	.....	.....	.....	.....	1	571.3	.....	1	.....	.....
565.6	.....	.....	.....	.....	1	.....	571.7	.....	1	.....	.....
566.2	.....	.....	.....	.....	2	.....	571.8	1	.....	.....	.....
566.4	.....	.....	.....	.....	1	.....	571.9	.....	.....	.....	1
567.0	.....	.....	.....	.....	1	.....	572.0	1	1	1	.....
567.1	.....	1	.....	.....	1	.....	572.6	.....	2	1	1
567.4	.....	.....	.....	.....	.....	2	572.8	1	.....	1	.....
567.5	.....	1	.....	.....	.....	.....	573.0	.....	1	.....	.....
567.6	.....	.....	.....	.....	1	1	573.2	1	2	.....	.....
567.8	1	.....	.....	.....	.....	.....	573.5	1	.....	.....	1
568.0	.....	.....	.....	.....	.....	1	573.6	.....	.....	1	.....
568.3	.....	1	.....	.....	.....	.....	573.7	.....	1	.....	.....
568.4	.....	1	.....	.....	.....	.....	573.8	.....	1	1	.....
568.5	.....	1	.....	.....	.....	.....	573.9	1	.....	2	1
568.6	.....	1	.....	.....	.....	.....	574.0	2	.....	2	1
568.7	1	.....	.....	1	.....	.....	574.1	1	.....	1	1
568.8	1	.....	.....	1	.....	.....	574.2	1	.....	.....	.....
569.1	.....	1	.....	.....	.....	.....	574.3	1	.....	1	1
569.2	.....	2	.....	.....	.....	.....	574.4	1	.....	.....	1
569.4	3	1	.....	.....	.....	.....	574.6	.....	.....	1	1
569.5	.....	1	1	.....	.....	.....	574.8	.....	.....	1	
569.6	.....	.....	.....	1	.....	.....	575.0	.....	.....	1	2
569.7	1	1	.....	1	.....	.....	575.5	.....	.....	.....	1
569.8	.....	.....	1	.....	.....	.....	575.7	.....	.....	.....	1
569.9	.....	.....	1	.....	.....	.....	576.0	.....	.....	.....	1
570.0	.....	.....	.....	2	.....	.....	.....	.....	.....	.....	.....
570.1	1		.....	.....	.....	.....	.....	.....	.....	.....	.....
570.2	1	.....	2	.....	.....	.....	.....	.....	.....	.....	.....
570.3	.....	.....	.....	1	.....	.....	.....	.....	.....	.....	.....
570.4	.....	.....	1	.....	.....	.....	.....	.....	.....	.....	.....
570.5	1	.....	.....	1	.....	.....	.....	.....	.....	.....	.....
570.6	2	.....	2	.....	.....	.....	.....	.....	.....	.....	.....
570.7	.....	.....	2	1	.....	.....	.....	.....	.....	.....	.....
570.8	.....	.....	.....	1	.....	.....	.....	.....	.....	.....	.....
570.9	.....	.....	1	.....	.....	.....	.....	.....	.....	.....	.....
571.0	.....	.....	.....	1	.....	.....	.....	.....	.....	.....	.....
571.3	.....	.....	.....	1	.....	.....	.....	.....	.....	.....	.....
Totals...	12	12	11	12	7	6		12	12	14	14

The presence of so much copper in this metal undoubtedly affected the actual assaying as well as the sampling of the bar.

In describing the taking of dip-samples I have spoken of the molten metal being "well mixed." This condition is absolutely essential to proper sampling. It is quite possible that in some of the cases already mentioned the molten metal had

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not been thoroughly mixed when the samples were taken. In the following case the molten metal certainly was not well mixed.

Two dip-samples were assayed in duplicate by me, showing a decided difference between the two samples. On a re-assay in duplicate, the same difference appeared. Each sample was also assayed once in two other laboratories. Table XXIII. summarizes the results:

TABLE XXIII.—*Differing Dip-Samples.*

	1st Sample.	2d Sample.
Bureau	615.6	612.8
	616.0	612.8
Re-assay	615.5	611.5
	616.3	611.1
2d laboratory	615.3	612.2
3d laboratory	615.5	611.9

Many melters, samplers, and assayers have an idea that unsatisfactory chip- and drill-samples can be corrected by simply melting the bar and taking fresh samples. There are, of course, cases where there was carelessness in the original melting and sampling, which can be cured by careful remelting and resampling, but in a vast majority of cases simple melting does little or no good. If there is a real difference of importance between two chips or two drillings or between chips and drillings of the same bar, it is due to the fact that the metal segregates on solidifying. This is a function of the chemical composition of the metal; and, under practical conditions, a remelting which does not materially change the composition can accomplish very little in reducing the segregation. If the chemical composition of the metal is such that it interferes with the actual assay-work, manifestly a remelting which does not change the composition of the bar will accomplish no good.

It therefore follows that when the assay-reports on an ordinary bar of miscellaneous bullion differ so widely as to be unsatisfactory, it should be melted and refined by strong oxidation. This will, of course, reduce the weight of the bar, but the loss in weight is nearly all base metal in most cases. The actual loss of gold will generally be small, except where a large amount, 50 oz. or more, of base must be removed, as in the case of many cyanide-bars, when the loss may become con-

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siderable. In the case already cited, the removal of 141.29 oz. from a bar weighing originally 643.30 oz. caused a loss of about 3.75 oz. of gold.

A bar carrying about 35 silver and weighing 494.26 oz. yielded varying assays on the gold, and was remelted with a loss of 0.63 oz. Again the assays were unsatisfactory, and it was remelted with a further loss of 1.09 oz. but without improvement in the assays. It was finally melted with a further loss of 5.20 oz. and still gave unsatisfactory assays. Two dip-samples were taken at each melting, and the eight samples were assayed in various service laboratories. Table XXIV. summarizes the results reported.

TABLE XXIV.—*Dip-Samples on Four Meltings.*

1st Melt.		2d Melt.		3d Melt.		4th Melt.	
Fineness.	Assays.	Fineness.	Assays.	Fineness.	Assays.	Fineness.	Assays.
806.0	1	809.9	1	812.8	1	818.3	1
806.1	1	810.1	2	813.0	1	818.4	1
806.3	2	810.6	3	813.2	2	818.8	1
806.4	2	810.7	1	813.3	1	819.1	1
806.5	1	810.8	2	813.6	2	819.2	1
806.7	1	810.9	1	813.7	1	819.3	1
806.8	2	811.0	2	813.8	1	819.5	1
806.9	3	811.1	1	813.9	1	819.9	1
807.0	2	811.3	1	814.2	1	820.2	2
807.1	1	811.6	2	814.6	2	820.5	3
807.2	3	811.8	2	814.8	2	820.6	1
807.4	2	812.2	1	814.9	1	820.8	1
807.5	2	812.3	1	815.0	3	821.0	1
807.6	3	.....	.....	815.3	1	821.3	1
807.7	2	.....	.....	.....	.....	821.4	2
807.8	3	.....	.....	.....	.....	.....	.....
807.9	1	.....	.....	.....	.....	.....	.....
808.0	1	.....	.....	.....	.....	.....	.....
808.1	2	.....	.....	.....	.....	.....	.....
808.2	3	.....	.....	.....	.....	.....	.....
808.3	1	.....	.....	.....	.....	.....	.....
808.5	2	.....	.....	.....	.....	.....	.....
Totals..	41		20		20		19

Another bar carrying about 35 silver and weighing 666.91 oz. was drilled at the ends and in the center, and the drills yielded the following results:

End Drills.		Center Drills.	
Fineness.	Assays.	Fineness.	Assays.
664.1	1	695.4	1
664.5	1	697.4	1
665.5	2	703.0	2

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This bar was remelted and two dip-samples were taken, while the cold bar was again drilled at the ends and in the center. After melting, however, even with the loss of 33.29 oz., the metal remained unsatisfactory, as shown by the following assays:

Dips.		Drills.			
		Ends.		Center.	
Fineness.	Assays.	Fineness.	Assays.	Fineness.	Assays.
705.5	1	699.5	2	706.7	1
706.6	1	699.9	1	707.9	1
706.8	1	701.3	1	708.9	1
707.8	1	701.6	1	725.0	1
707.9	1	701.8	1	726.3	1
708.3	1			727.0	1
708.5	1				
708.7	1				

From these gold-assays the silver cannot be given with any exactness, but it was probably about 40.

Conclusions.

The foregoing data are summarized in the following conclusions:

In sampling deposits of miscellaneous gold-bullion weighing more than 300 oz.:

There are various cases where either a chip- or a drill-sample may be satisfactory.

There are various cases where a drill-sample is better than a chip-sample.

Where the assayer is acquainted with the metal, he may accept a chip- or a drill-sample.

On an unknown bullion it is unsafe to accept any sample except a properly prepared dip-sample.

In many cases, particularly of cyanide-bullion, the composition of the metal interferes with the actual assaying and the bullion must be refined before one can expect to determine the gold accurately.

Discussion.

Edward Keller, Perth Amboy, N. J. (communication to the Secretary *):—Mr. Dewey, in his paper, offers a liberal addition of new data to those already to be found in the technical literature on this subject. However, now as before, the work issuing from the U. S. mints and assay-offices conveys the

* Received Jan. 23, 1913.

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impression that systematic sampling in those institutions is more or less a secondary consideration, the chief attention being given to the principle of producing approximately homogeneous bars; recourse being taken to remelting or refining the bullion. This practice has been definitely rejected in the realm of copper-bullion sampling, for the reason that in such operations there are always unaccounted losses. Does not the same objection hold true for gold-bullions; although, perhaps, in a lesser degree?

Differences of opinion would seem justified on the following statements made by Mr. Dewey:

1. "Only two classes of alloys can form homogeneous bars—those that are solid solutions, and the eutectics."

There is no doubt as to this characteristic of the eutectics. As to the solid solutions, the late eminent French metallurgist, Floris Osmond,[6] demonstrated conclusively that silver remains in solid solution with copper up to the amount of 1 per cent.; yet silver-bullions, as well as pure copper-silver alloys, within the mentioned range of composition show a very marked degree of diffusion and the bars a consequent heterogeneity, in this case improperly called segregation. Solid solutions are homogeneous only in their microstructure, showing on etched surfaces under the microscope but a single structural element. From the freezing-point curves, theoretically established by Bakhuis Roozeboom,[7] we may deduce the fact that even such binary alloys, the whole series of which form solid solutions, and to which the gold-silver series belong, are bound to have diffusion during freezing or solidification and, therefore, to show heterogeneity in chemical composition in their solid state, although the degree of the latter may be much less than in some alloys of another class. Fig. 1 shows the general diagram of freezing-points of the class of alloys to which the gold-silver series belong. The length of the ordinates at C and at D represents the temperature of the freezing-points of the two metals A and B. Of the two curves connecting C and D, the upper represents the initial and the lower the final freezing-points of the alloy-series. Between the two curves lies the range of the freezing-temperature. The

[6] Sur les alliages du groupe argent-cuivre, *Comptes rendus de l'Académie des Sciences*, vol. cxxiv., p. 1234 (1897).

[7] Erstarrungs punkte der Mischkrystalle zweier Stoffe, *Zeitschrift für physikalische Chemie*, vol. xxx., p. 385 (1899).

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greater the distance between the two curves, the greater the difference between the composition of the initially-freezing and the finally-freezing alloy, or the greater the diffusion and the heterogeneity of the solid alloy. There seems to exist no satisfactory freezing-point curve of the gold-silver series of alloys; one by Roberts-Austen and Kirke Rose consists of few determined points, but it ignores the final freezing.

Mr. Dewey omits to mention that a chemical compound would also be a possibility for a homogeneous alloy. A. Levol[8] first drew attention to this, although he mistook a eutectic for a chemical compound; in his time the character of the eutectic had not yet been demonstrated. Numerous instances of chem-

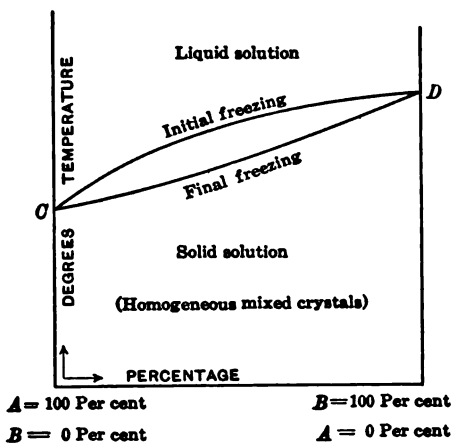

FIG. 1.—FREEZING-POINT DIAGRAM AFTER BAKHUIS ROOZEBOOM.

ical compounds are now known among strictly metallic alloys. I would only refer to the important series of copper-tin and copper-aluminium. The possibility of a gold-bullion being a chemical compound is very remote; unless that some day aluminium be employed as a collecting medium.

2. "It is a common practice in the Mint Service to drill half-way through a bar at diagonally opposite corners of the top, and unite the drillings for a top-sample. The remaining corners are drilled half-way through from the bottom, and the drillings are mixed for the bottom-sample. Occasionally, especially when sampling very base bars, the four drillings are kept separate; and sometimes holes are drilled near the center of the bar also."

[8] Mémoire sur les alliages, considérés sous le rapport de leur composition chimique, *Annales de Chimie et de Physique*, Third Series, vol. xxxvi., p. 194 (1852).

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Naturally, if the bars be homogeneous, it is of little importance where and how deep the holes are drilled; but, if there has been diffusion or the bars show "segregation," the corners of the bar would appear the worst possible positions to drill for a correct sample; for these positions are the well-demonstrated centers of concentration—or, as many insist, segregation. From Mr. Dewey's description it does not appear clearly whether these samples are taken as the real sample of the bars, or, perhaps, to show their heterogeneity, and when the latter is established refining is resorted to.

The shape of a bar is always an important factor in the development of heterogeneity, and for that reason the lack of any information on that point may be considered a deplorable omission on the part of Mr. Dewey.

I have but recently endeavored, by graphical demonstration, to show the irrationality of drilling in any one place a bar of bullion with an approximately square cross-section, and I have shown why a saw-section of such a bar would yield a correct sample.[9] Why could not the same scientific principles be applied to the sampling of heterogeneous gold-bullion bars?

Most of the objectionable features of sampling a bar by drilling are eliminated in the comparatively thin plate; this I recommended about 17 years ago in place of the time-honored copper-pig (bar), and the plate, or slab, is now almost universally adopted by the copper-smelteries. While it is not a cure for all evils in sampling, it gives commercially very satisfactory results, and it has finally eliminated the formerly often-recurring sampling-controversies.

3. "Drill-samples of brittle bars are, however, liable to be unsatisfactory, because the fine and coarse portions may differ considerably in composition."

In copper-bullion samples, the fine portion is generally higher in content of precious metals and lower in copper than the coarse; but this is not deemed a difficulty, since the sample is separated by means of a sieve into coarse and fine parts, and the two are weighed in their proper ratio for the assay-charge. Grading into three or more sizes and weighing in proportioned quantities would add but little difficulty.

[9] The Mathematics of Copper Sampling, *Engineering and Mining Journal*, vol. xciii., p. 703 (Apr. 6, 1912).

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Temperature Conversion Tables.

BY LEONARD WALDO, NEW YORK, N. Y.

(New York Meeting, February, 1911.)

THE recent and rapid development of the physics of engineering materials at temperatures as low as that of liquid air and as high as that of the electric arc, has drawn renewed attention to the absence of tables, properly printed and spaced, for the convenient and accurate translation of temperature from centigrade to Fahrenheit scales and *vice versa*. It is to be noted that the Fahrenheit scale is the almost universal usage in foundries and shops in the United States, though the laboratories of the same works employ the centigrade notation. The growing use of the electric furnace in Germany and France calls for much American citation all in centigrade high temperatures, and while the arithmetical calculation is simple, it becomes burdensome when often repeated.

I have been surprised at the numerous reprints of Table I. from centigrade to Fahrenheit degrees, first privately printed for personal use, which have been made by works, blue-prints, text-books, and technical journals, and I hope that Table II., from Fahrenheit to centigrade degrees, will be equally useful.

For mental calculation the easiest rule to convert centigrade into Fahrenheit is to add 32° to twice the centigrade reading less one-tenth of this product:

$$\text{Thus } 1000° \text{ C.} = 2000° - 200° + 32° = 1832° \text{ F.},$$

and to convert Fahrenheit to centigrade, subtract 32° from the Fahrenheit reading, multiply by 10 and divide by 2 and 9 successively:

$$\text{Thus } 1000° \text{ F.} = 1000° - 32° = 968° \times 10 = 9680° \div 2 = 4840° \div 9 = 537.7\dot{7}° \text{ C.}$$

Another and often very convenient method is by use of a graphic construction as follows:

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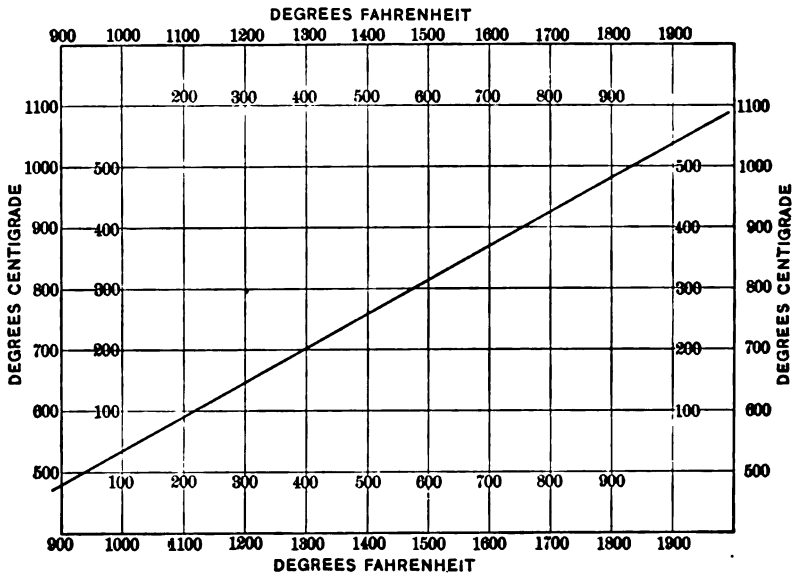

FIG. 1.—HEAT-TREATING RANGES, CENTIGRADE-FAHRENHEIT CONVERSION SCALE.

This construction may be limited to a small section, and for use it may be drawn on accurately spaced cross-section paper showing one space for each degree for both co-ordinates.

The arguments of degrees Fahrenheit and degrees centigrade may be repeated under each other in cycles of 900° F., and extended sideways for each 500° C. Thus the arguments under 100° F. of the diagram given would be:

100°
1000°
1900°
2800°, etc.,

and to the left of the argument 100° C., the numbers would be:

2100° 1600° 1100° 600° 100°,

and so for each line of the centigrade degrees argument.

When such a graphic construction is made, the boundary lines should be heavy and distinctly indicate the channel in which the corresponding lines of numbers lie. The eye readily follows such a guidance and there is practically no confusion, while the compactness of the graphic chart makes it most convenient. For accurate work over limiting temperatures, such a diagram may be enlarged to single degrees for each space of the cross-section paper, always preserving the 5 : 9 ratio between the two co-ordinates.

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TABLE I.—*Centigrade and Fahrenheit Temperature Conversion Table.*

C.°	0	10	20	30	40	50	60	70	80	90
	F.	F.	F.	F.	F.	F.	F.	F.	F.	F.
—200	—328	—346	—364	—382	—400	—418	—436	—454	.....	.....
—100	—148	—166	—184	—202	—220	—238	—256	—274	—292	—310
—0	+32	+14	—4	—22	—40	—58	—76	—94	—112	—130
0	32	50	68	86	104	122	140	158	176	194
100	212	230	248	266	284	302	320	338	356	374
200	392	410	428	446	464	482	500	518	536	554
300	572	590	608	626	644	662	680	698	716	734
400	752	770	788	806	824	842	860	878	896	914
500	932	950	968	986	1004	1022	1040	1058	1076	1094
600	1112	1130	1148	1166	1184	1202	1220	1238	1256	1274
700	1292	1310	1328	1346	1364	1382	1400	1418	1436	1454
800	1472	1490	1508	1526	1544	1562	1580	1598	1616	1634
900	1652	1670	1688	1706	1724	1742	1760	1778	1796	1814
1000	1832	1850	1868	1886	1904	1922	1940	1958	1976	1994
1100	2012	2030	2048	2066	2084	2102	2120	2138	2156	2174
1200	2192	2210	2228	2246	2264	2282	2300	2318	2336	2354
1300	2372	2390	2408	2426	2444	2462	2480	2498	2516	2534
1400	2552	2570	2588	2606	2624	2642	2660	2678	2696	2714
1500	2732	2750	2768	2786	2804	2822	2840	2858	2876	2894
1600	2912	2930	2948	2966	2984	3002	3020	3038	3056	3074
1700	3092	3110	3128	3146	3164	3182	3200	3218	3236	3254
1800	3272	3290	3308	3326	3344	3362	3380	3398	3416	3434
1900	3452	3470	3488	3506	3524	3542	3560	3578	3596	3614
2000	3632	3650	3668	3686	3704	3722	3740	3758	3776	3794
2100	3812	3830	3848	3866	3884	3902	3920	3938	3956	3974
2200	3992	4010	4028	4046	4064	4082	4100	4118	4136	4154
2300	4172	4190	4208	4226	4244	4262	4280	4298	4316	4334
2400	4352	4370	4388	4406	4424	4442	4460	4478	4496	4514
2500	4532	4550	4568	4586	4604	4622	4640	4658	4676	4694
2600	4712	4730	4748	4766	4784	4802	4820	4838	4856	4874
2700	4892	4910	4928	4946	4964	4982	5000	5018	5036	5054
2800	5072	5090	5108	5126	5144	5162	5180	5198	5216	5234
2900	5252	5270	5288	5306	5324	5342	5360	5378	5396	5414
3000	5432	5450	5468	5486	5504	5522	5540	5558	5576	5594
3100	5612	5630	5648	5666	5684	5702	5720	5738	5756	5774
3200	5792	5810	5828	5846	5864	5882	5900	5918	5936	5954
3300	5972	5990	6008	6026	6044	6062	6080	6098	6116	6134
3400	6152	6170	6188	6206	6224	6242	6260	6278	6296	6314
3500	6332	6350	6368	6386	6404	6422	6440	6458	6476	6494
3600	6512	6530	6548	6566	6584	6602	6620	6638	6656	6674
3700	6692	6710	6728	6746	6764	6782	6800	6818	6836	6854
3800	6872	6890	6908	6926	6944	6962	6980	6998	7016	7034
3900	7052	7070	7088	7106	7124	7142	7160	7178	7196	7214
C.°	0	10	20	30	40	50	60	70	80	90

C.°	F.°
1	1.8
2	3.6
3	5.4
4	7.2
5	9.0
6	10.8
7	12.6
8	14.4
9	16.2
10	18.0

F.°	C.°
1	.56
2	1.11
3	1.67
4	2.22
5	2.78
6	3.33
7	3.89
8	4.44
9	5.00
10	5.56
11	6.11
12	6.67
13	7.22
14	7.78
15	8.33
16	8.89
17	9.44
18	10.00

EXAMPLES: 1347° C. = 2444° F. + 12.6° F. = 2456.6° F. 3367° F. = 1850° C. + 2.78° C. = 1852.78° C.

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TABLE II.—*Fahrenheit and Centigrade Temperature Conversion Table.*

Dots over figures indicate recurring decimals.

F.°	0	10	20	30	40	50	60	70	80	90
	C.	C.	C.	C.	C.	C.	C.	C.	C.	C.
—400	—240.0	—245.5	—251.1	—256.6	—262.2	—267.7	.....	.....	.....	.....
—300	—184.4	—190.0	—195.5	—201.1	—206.6	—212.2	—217.7	—223.3	—228.8	—234.4
—200	—128.8	—134.4	—140.0	—145.5	—151.1	—156.6	—162.2	—167.7	—173.3	—178.8
—100	—73.3	—78.8	—84.4	—90.0	—95.5	—101.1	—106.6	—112.2	—117.7	—123.3
—0	—17.7	—23.3	—28.8	—34.4	—40.0	—45.5	—51.1	—56.6	—62.2	—67.7
0	—17.7	—12.2	—6.6	—1.1	+4.4	+10.0	+15.5	+21.1	+26.6	+32.2
100	37.7	43.3	48.8	54.4	60.0	65.5	71.1	76.6	82.2	87.7
200	93.3	98.8	104.4	110.0	115.5	121.1	126.6	132.2	137.7	143.3
300	148.8	154.4	160.0	165.5	171.1	176.6	182.2	187.7	193.3	198.8
400	204.4	210.0	215.5	221.1	226.6	232.2	237.7	243.3	248.8	254.4
500	260.0	265.5	271.1	276.6	282.2	287.7	293.3	298.8	304.4	310.0
600	315.5	321.1	326.6	332.2	337.7	343.3	348.8	354.4	360.0	365.5
700	371.1	376.6	382.2	387.7	393.3	398.8	404.4	410.0	415.5	421.1
800	426.6	432.2	437.7	443.3	448.8	454.4	460.0	465.5	471.1	476.6
900	482.2	487.7	493.3	498.8	504.4	510.0	515.5	521.1	526.6	532.2
1000	537.7	543.3	548.8	554.4	560.0	565.5	571.1	576.6	582.2	587.7
1100	593.3	598.8	604.4	610.0	615.5	621.1	626.6	632.2	637.7	643.3
1200	648.8	654.4	660.0	665.5	671.1	676.6	682.2	687.7	693.3	698.8
1300	704.4	710.0	715.5	721.1	726.6	732.2	737.7	743.3	748.8	754.4
1400	760.0	765.5	771.1	776.6	782.2	787.7	793.3	798.8	804.4	810.0
1500	815.5	821.1	826.6	832.2	837.7	843.3	848.8	854.4	860.0	865.5
1600	871.1	876.6	882.2	887.7	893.3	898.8	904.4	910.0	915.5	921.1
1700	926.6	932.2	937.7	943.3	948.8	954.4	960.0	965.5	971.1	976.6
1800	982.2	987.7	993.3	998.8	1004.4	1010.0	1015.5	1021.1	1026.6	1032.2
1900	1037.7	1043.3	1048.8	1054.4	1060.0	1065.5	1071.1	1076.6	1082.2	1087.7
2000	1093.3	1098.8	1104.4	1110.0	1115.5	1121.1	1126.6	1132.2	1137.7	1143.3
2100	1148.8	1154.4	1160.0	1165.5	1171.1	1176.6	1182.2	1187.7	1193.3	1198.8
2200	1204.4	1210.0	1215.5	1221.1	1226.6	1232.2	1237.7	1243.3	1248.8	1254.4
2300	1260.0	1265.5	1271.1	1276.6	1282.2	1287.7	1293.3	1298.8	1304.4	1310.0
2400	1315.5	1321.1	1326.6	1332.2	1337.7	1343.3	1348.8	1354.4	1360.0	1365.5
2500	1371.1	1376.6	1382.2	1387.7	1393.3	1398.8	1404.4	1410.0	1415.5	1421.1
2600	1426.6	1432.2	1437.7	1443.3	1448.8	1454.4	1460.0	1465.5	1471.1	1476.6
2700	1482.2	1487.7	1493.3	1498.8	1504.4	1510.0	1515.5	1521.1	1526.6	1532.2
2800	1537.7	1543.3	1548.8	1554.4	1560.0	1565.5	1571.1	1576.6	1582.2	1587.7
2900	1593.3	1598.8	1604.4	1610.0	1615.5	1621.1	1626.6	1632.2	1637.7	1643.3
3000	1648.8	1654.4	1660.0	1665.5	1671.1	1676.6	1682.2	1687.7	1693.3	1698.8
3100	1704.4	1710.0	1715.5	1721.1	1726.6	1732.2	1737.7	1743.3	1748.8	1754.4
3200	1760.0	1765.5	1771.1	1776.6	1782.2	1787.7	1793.3	1798.8	1804.4	1810.0
3300	1815.5	1821.1	1826.6	1832.2	1837.7	1843.3	1848.8	1854.4	1860.0	1865.5
3400	1871.1	1876.6	1882.2	1887.7	1893.3	1898.8	1904.4	1910.0	1915.5	1921.1
3500	1926.6	1932.2	1937.7	1943.3	1948.8	1954.4	1960.0	1965.5	1971.1	1976.6
3600	1982.2	1987.7	1993.3	1998.8	2004.4	2010.0	2015.5	2021.1	2026.6	2032.2
F.°	0	10	20	30	40	50	60	70	80	90

F.°	C.°
1	0.5
2	1.1
3	1.6
4	2.2
5	2.7
6	3.3
7	3.8
8	4.4
9	5.0

EXAMPLES: —246.0° F. = —151.11° C. — 3.33° C. = —154.44° C.
3762° F. = 2071.11° C. + 1.11° C. = 2072.22° C.
2423.5° F. = 1326.666° C. + 1.666° C. + 0.277° C. = 1328.609° C.

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TABLE II.—*Fahrenheit and Centigrade Temperature Conversion Table—Continued.*

Dots over figures indicate recurring decimals.

F.°	0	10	20	30	40	50	60	70	80	90
	C.	C.	C.	C.	C.	C	C.	C.	C.	C.
3700	2037.7	2043.3	2048.8	2054.4	2060.0	2065.5	2071.1	2076.6	2082.2	2087.7
3800	2093.3	2098.8	2104.4	2110.0	2115.5	2121.1	2126.6	2132.2	2137.7	2143.3
3900	2148.8	2154.4	2160.0	2165.5	2171.1	2176.6	2182.2	2187.7	2193.3	2198.8
4000	2204.4	2210.0	2215.5	2221.1	2226.6	2232.2	2237.7	2243.3	2248.8	2254.4
4100	2260.0	2265.5	2271.1	2276.6	2282.2	2287.7	2293.3	2298.8	2304.4	2310.0
4200	2315.5	2321.1	2326.6	2332.2	2337.7	2343.3	2348.8	2354.4	2360.0	2365.5
4300	2371.1	2376.6	2382.2	2387.7	2393.3	2398.8	2404.4	2410.0	2415.5	2421.1
4400	2426.6	2432.2	2437.7	2443.3	2448.8	2454.4	2460.0	2465.5	2471.1	2476.6
4500	2482.2	2487.7	2493.3	2498.8	2504.4	2510.0	2515.5	2521.1	2526.6	2532.2
4600	2537.7	2543.3	2548.8	2554.4	2560.0	2565.5	2571.1	2576.6	2582.2	2587.7
4700	2593.3	2598.8	2604.4	2610.0	2615.5	2621.1	2626.6	2632.2	2637.7	2643.3
4800	2648.8	2654.4	2660.0	2665.5	2671.1	2676.6	2682.2	2687.7	2693.3	2698.8
4900	2704.4	2710.0	2715.5	2721.1	2726.6	2732.2	2737.7	2743.3	2748.8	2754.4
5000	2760.0	2765.5	2771.1	2776.6	2782.2	2787.7	2793.3	2798.8	2804.4	2810.0
5100	2815.5	2821.1	2826.6	2832.2	2837.7	2843.3	2848.8	2854.4	2860.0	2865.5
5200	2871.1	2876.6	2882.2	2887.7	2893.3	2898.8	2904.4	2910.0	2915.5	2921.1
5300	2926.6	2932.2	2937.7	2943.3	2948.8	2954.4	2960.0	2965.5	2971.1	2976.6
5400	2982.2	2987.7	2993.3	2998.8	3004.4	3010.0	3015.5	3021.1	3026.6	3032.2
5500	3037.7	3043.3	3048.8	3054.4	3060.0	3065.5	3071.1	3076.6	3082.2	3087.7
5600	3093.3	3098.8	3104.4	3110.0	3115.5	3121.1	3126.6	3132.2	3137.7	3143.3
5700	3148.8	3154.4	3160.0	3165.5	3171.1	3176.6	3182.2	3187.7	3193.3	3198.8
5800	3204.4	3210.0	3215.5	3221.1	3226.6	3232.2	3237.7	3243.3	3248.8	3254.4
5900	3260.0	3265.5	3271.1	3276.6	3282.2	3287.7	3293.3	3298.8	3304.4	3310.0
6000	3315.5	3321.1	3326.6	3332.2	3337.7	3343.3	3348.8	3354.4	3360.0	3365.5
6100	3371.1	3376.6	3382.2	3387.7	3393.3	3398.8	3404.4	3410.0	3415.5	3421.1
6200	3426.6	3432.2	3437.7	3443.3	3448.8	3454.4	3460.0	3465.5	3471.1	3476.6
6300	3482.2	3487.7	3493.3	3498.8	3504.4	3510.0	3515.5	3521.1	3526.6	3532.2
6400	3537.7	3543.3	3548.8	3554.4	3560.0	3565.5	3571.1	3576.6	3582.2	3587.7
6500	3593.3	3598.8	3604.4	3610.0	3615.5	3621.1	3626.6	3632.2	3637.7	3643.3
6600	3648.8	3654.4	3660.0	3665.5	3671.1	3676.6	3682.2	3687.7	3693.3	3698.8
6700	3704.4	3710.0	3715.5	3721.1	3726.6	3732.2	3737.7	3743.3	3748.8	3754.4
6800	3760.0	3765.5	3771.1	3776.6	3782.2	3787.7	3793.3	3798.8	3804.4	3810.0
6900	3815.5	3821.1	3826.6	3832.2	3837.7	3843.3	3848.8	3854.4	3860.0	3865.5
7000	3871.1	3876.6	3882.2	3887.7	3893.3	3898.8	3904.4	3910.0	3915.5	3921.1
7100	3926.6	3932.2	3937.7	3943.3	3948.8	3954.0	3960.0	3965.5	3971.1	3976.6
7200	3982.2	3987.7	3993.3	3998.8	4004.4	4010.0	4015.5	4021.1	4026.6	4032.2
7300	4037.7	4043.3	4048.8	4054.4	4060.0	4065.5	4071.1	4076.6	4082.2	4087.7
7400	4093.3	4098.8	4104.4	4110.0	4115.5	4121.1	4126.6	4132.2	4137.7	4143.3
7500	4148.8	4154.4	4160.0	4165.5	4171.1	4176.6	4182.2	4187.7	4193.3	4198.8
7600	4204.4	4210.0	4215.5	4221.1	4226.6	4232.2	4237.7	4243.3	4248.8	4254.4
7700	4260.0	4265.5	4271.1	4276.6	4282.2	4287.7	4293.3	4298.8	4304.4	4310.0
7800	4315.5	4321.1	4326.6	4332.2	4337.7	4343.3	4348.8	4354.4	4360.0	4365.5
7900	4371.1	4376.6	4382.2	4387.7	4393.3	4398.8	4404.4	4410.0	4415.5	4421.1
F.°	0	10	20	30	40	50	60	70	80	90

F.°	C.°
1	0.5
2	1.1
3	1.6
4	2.2
5	2.7
6	3.3
7	3.8
8	4.4
9	5.0

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Discussion.

Herbert Haas, San Francisco, Cal. (communication to the Secretary *):—Referring to Leonard Waldo's paper on Temperature Conversion Tables, I believe the following formulas can be easily remembered, and by reference to the thermometer scales of Fahrenheit (F.), Celsius (C.), and Réaumur (R.), their derivation and relation are at once understood:

$$n^\circ \text{ F.} = \tfrac{4}{9}(n^\circ - 32) \text{ R.} = \tfrac{5}{9}(n^\circ - 32) \text{ C.}$$
$$n^\circ \text{ C.} = (\tfrac{9}{5}\, n^\circ + 32) \text{ F.} = \tfrac{4}{5}\, n^\circ \text{ R.}$$
$$n^\circ \text{ R.} = (\tfrac{9}{4}\, n^\circ + 32) \text{ F.} = \tfrac{5}{4}\, n^\circ \text{ C.}$$

As the freezing point of water is at 32° F., corresponding to 0° C. and 0° R., 32 must be deducted from the Fahrenheit degrees when these are converted into C. or R. degrees. Likewise, to C. or R. degrees converted into F. degrees 32 must be added. The fractions $\frac{4}{9}$ and $\frac{5}{9}$ and their reciprocals are obtained thus:

$$212^\circ \text{ F.} - 32^\circ \text{ F.} = 180; \quad \frac{180}{100} = \tfrac{9}{5} \text{ and } \frac{100}{180} = \tfrac{5}{9}.$$

As 80° R. corresponds to 100° C., and to 212° F., these fractions become $\frac{9}{4}$ and $\frac{4}{9}$. Examples:

$$109.5^\circ \text{ F.} = \tfrac{4}{9}(109.5 - 32) = \tfrac{4}{9} \times 77.5 = 34.444 \text{ R.}$$
$$= \tfrac{5}{9}(109.5 - 32) = \tfrac{5}{9} \times 77.5 = 43.1 \text{ C.}$$
$$32^\circ \text{ F.} = \tfrac{5}{9}(32 - 32) = \tfrac{5}{9} \times 0 = 0^\circ \text{ C.}$$
$$-55^\circ \text{ C.} = \tfrac{4}{5} \times -55 = -44^\circ \text{ R.}$$
$$= \tfrac{9}{5} \times -55 + 32 = \frac{-495}{5} + 32 = -99 + 32 = -67 \text{ F.}$$
$$0^\circ \text{ F.} = \tfrac{5}{9}(0^\circ \text{ F.} - 32^\circ \text{ F.}) = \tfrac{5}{9} \times -32 = \frac{-160}{9}$$
$$= -17.7^\circ \text{ C.}$$

* Received May 5, 1913.

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DISCUSSIONS.

Discussions of papers contained in this volume have been placed immediately after the respective papers with the exception of those here printed, which were received too late to be printed in connection with the papers to which they relate.

Clinton Iron-Ore Deposits in Kentucky and Tennessee.

Discussion of the paper of S. Whinery, p. 25.

GRAHAM MACFARLANE, Louisville, Ky. (communication to the Secretary*):—In the paper by S. Whinery under the above caption mention is made of the occurrence of Clinton iron-ore in Kentucky and Tennessee. I believe that this is really a ferruginous limestone in what the Tennessee Geological Survey calls the Fernvale formation. In Davidson and Sumner counties, Tennessee, this occurs at about the horizon stated by Mr. Whinery. The maximum thickness of the deposit is from 10 to 12 ft., but it averages between 6 and 8 ft., lying practically horizontal.

The Red River Furnace Co., under my direction, opened a mine in the Fernvale "red rock" near Goodlettsville, Tenn., several years ago. When it was first found we assumed it to be Clinton ore, but the absence of the characteristic Clinton fossiliferous structure and other evidence satisfied us that it is really limestone saturated with iron oxide.

I think the very low ratio of SiO_2 and Al_2O_3 to iron content as compared with the usual ratio in the Clinton ore is also evidence in favor of its being the Fernvale. The Tennessee Geological Survey states that at outcrops where this ore has been leached it carries about 44 per cent. of iron; under cover, from 20 to 25 per cent. of iron and from 3 to 6 per cent. of silica, the remainder being chiefly calcium carbonate. Its phosphorus content is irregular and generally rather high for a foundry-iron mixture. Since it is more than self-fluxing, we have at times operated with a mixture of brown hematite and red ore with no limestone.

* Received June 30, 1913.

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Development of the American Water-Jacket Lead Blast-Furnace.

Discussion of the paper of R. C. Canby, p. 736.

FRANCIS DRAKE, Bulawayo, Rhodesia, So. Africa (communication to the Secretary *):—I should like to place on record some data in addition to those given by Mr. Canby in his paper. I regret that my notes have been destroyed so that I can give no accurate measurements of the water jacket that I am about to describe.

This furnace was erected in Cerro Gordo, Inyo county, Cal., early in 1875, and I think it quite possible that it was the first applied in lead smelting. It was made of plate iron and elliptic in shape, the inner dimensions being about 54 by 42 in. by 48 in. high. The distance between the inner and outer plates was about 6 in. Tuyere holes, one at the back and two on each side, about 12 in. in diameter, were made in this, and in these were placed cast-iron water tuyeres about 16 in. long, the diameter of the blast orifice at the point being about 3 in. The height from tuyeres to feed door was about 9 ft., but the top of the charge was kept about 5 ft. above the tuyeres. There was a space of about 12 in. between the jacket and the top brick-work. This brick-work was supported by a cast-iron plate resting on four cast-iron columns. At the beginning, the jacket was lined with air-dried radial bricks, made of crushed quartz and fire-clay, as it was thought that the jacket would not stand the heat on its bare plates. After a short time the practice was altered and no lining was put in. The jacket was closed at the top and had a steam drum connected by two pipes at the back. This steam drum was connected to the boiler in which the steam was made for driving the Baker blower, the rock breaker and the pumps. The bottom of the jacket was also connected to the boiler. The jacket was run under boiler pressure (80 lb.) for some time, but eventually cracks developed in the inner plate between the tuyere holes,

* Received July 7, 1913.

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and water was allowed to run through it from the service tank which supplied the tuyeres.

This furnace produced as much as 13 tons of silver-lead bullion per 24 hr. for days at a time. The tonnage of ore put through was unknown. The fuel was charcoal.

The design of the furnace was due to M. W. Belshaw, the owner, and William Goodrum, metallurgist, and it was made in San Francisco.

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INDEX.

[NOTE.—In this Index the names of authors of papers are printed in small capitals, and the titles of papers in italics. References to papers expressly treating of the subject named are likewise in italics; and casual notices, giving but little information, are usually indicated by bracketed page-numbers. The titles of papers presented, but not printed in this volume, are followed by bracketed page-numbers only.]

ABBOTT, ROBERT R.: *The Action of Various Commercial Carbonizing-Materials*, viii, 371–421.

Discussions: on Alloys of Cobalt with Chromium and Other Metals, 578.

on Note on the Case-Hardening of Special Steels, 428.

Action of Various Commercial Carbonizing-Materials (ABBOTT), viii, 371–420: *Discussion* (ABBOTT), 421; (HIBBARD), 421.

Adams iron-mine, Cuyuna district, Minn., 11.

Allen, E. T., and Day, Arthur L.: melting-points of minerals, [135].

Alloys:

aluminum-titanium, 293.

bibliography, 849.

bronzes, 847.

composition of (tabulations), 827, 835, 845.

copper-tin, 835.

copper-tin-zinc, 847.

effect of titanium in, 292.

ferro-titanium: manufacture, 293.

gold-silver: sampling for assay, 853–882.

lead-antimony, 827.

lead-antimony-tin, 830.

lead-tin, 828.

Metallography, 825–850.

microstructure, 833.

nickel-steel: case-hardening, 422–429.

phosphor-bronze, 848.

stellite (chromium-cobalt-tungsten), 574.

tin-antimony-copper, 839.

tin-antimony-zinc, 846.

titanium-steel, 292.

zinc-aluminum, 846.

zinc-antimony-copper, 846.

zinc-tin, 846.

Alloys of Chromium with Cobalt and Other Metals (HAYNES), ix, 573–577; *Discussion* (ABBOTT), 578; (HAYNES), 577–579; (JOHNSON), 579; (LEDOUX), 579; (MOORE), 577; (WELLMAN), 578; (ZAPFFE), 577.

Alumina: effect in blast-furnace slags, 123–142.

Aluminium: effect of alloying with titanium, 293.

Aluminium-zinc alloys: metallography, 846.

Alundum Extraction Thimble Used in the Determination of Copper (BAHNEY), xii, 851–852.

Digitized by Google

AMBERG, RICHARD: *The Function of Slag in Electric Steel-Refining*, xii, 557–568.
American Smelters Securities Co.: reverberatory copper-furnace, Garfield, Utah, 804.
American Smelting & Refining Co.: bag-filtration plants, 708–735.
AMMON, MARK A.: *Measurements and Relations of Hardness and Depth of Carbonization in Case-Hardened Steel*, viii, 356–369.
Ammonia: recovery from coke-oven gases, 167, 174, 178–180.
Anaconda Copper Mining Co., Anaconda, Mont.:
  chemistry of reduction processes, 806–817.
  reverberatory furnace data, 786.
Analyses:
  coal, Diamondville, Wyo., 784.
  copper-furnace charges, 788, 790.
  copper-furnace gases, Kyshtim, Russia, 788.
  copper-mattes, 731, 788.
  copper-ores, 821–823.
  copper-slags, 755, 773–776, 788, 790.
  fuels: reveberatory furnace, 788.
  gases: fuel, Great Falls, Mont., copper-smelter, 788.
    lead-furnace, 723, 728, 732.
    producer, from wood, Kyshtim, Russia, 788.
    copper-furnace, Kyshtim, Russia, 788.
  iron: charcoal, 339, 341, 494.
    ingot, 494.
    puddled, 494.
  iron-ore: Clinton, Kentucky, 26.
    Cuyuna district, Minn., 8, 9.
    Ta-yeh, China, 32.
    titaniferous, 23.
  iron-ore concentrates, Sellwood, Ont., 52, 54.
  lead-furnace fume, 723, 728, 732.
  nickel steel, 357.
  rocks: gabbro, Boulder county, Colo., 16.
    quartz-monzonite, 807.
  steel, 357, 494.
Analytical methods:
  for iron-ore, 22.
  for pig-iron: U. S. Steel Corporation, 220–249.
  for rocks, [16].
ANDREWS, C. W.: *By-Product Coke*, ix, 181–183.
Annealing: influence on mechanical properties of low-carbon steel, 513–547.
Antimony-lead-tin alloys: metallography, 826.
Antimony-tin-copper alloys: metallography, 834.
Arsenic: recovery from lead-furnace fume, 732.
Assaying: sampling of gold-bullion, 853–882.
Assays: copper-slags, Washoe smelter, Anaconda, Mont., 784.

Bag-filtration plants, 708–735.
BAHNEY, L. W.: *The Alundum Extraction-Thimble Used in the Determination of Copper*, xiii, 851–852.
Balbach process for desilvering lead, 741.
Ball, Sydney H.: dike form of iron-ore deposits, Wyoming, [24].
Balliot cupola, [210].
BARROWS, W. A., JR., and ZAPFFE, CARL: *Iron-Ores of the South Range of the Cuyuna District, Minn.*, viii, 3–13.
Barrows iron-mine, Cuyuna district, Minn., 11.

Digitized by Google

Basic open-hearth steel (see also Steel practice); preparing for castings, 263–268.
Bee-hive coke-ovens:
comparison with other types, 157, 179.
cost per oven, 158.
Bibliographies:
alloys: metallography, 849.
titanium: history, occurrence, use in iron metallurgy, 306.
Blast-furnace practice:
blowing in, 105–122.
coke: structure desired, 155.
Dutch oven for drying out, 107.
Johnson slag-tester, 139.
new type of furnace, 91–104.
ores: Cuyuna district, Minn., 12.
slags: composition, 128.
effect of alumina, 123–142.
effect of lime, 125.
effect of magnesia, 126.
influence of temperature of formation, 138.
ratios of lime, silica, and alumina, 128.
utility of efficiency-records, 143–153.
Blast-furnace reactions (copper), 813.
Blast-furnaces (lead): *development of the water-jacket*, 736–740.
BLAUVELT, WILLIAM HUTTON: *The Manufacture of Coke*, ix, 154–169, 184, 185, 193–196.
Discussion on Recent Developments in the Inspection of Steel Rails, 278.
Blowing-In a Blast-Furnace (SWEETSER), ix, 105–112; *Discussion* (FIRMSTONE), 115–119; (HOWARD), 113–115; (JOHNSON), 112–113; (LEE), 119–121; (SWEETSER), 121–122.
Bolivia: mineral wealth and mining conditions, 640.
BORDEAUX, ALBERT F. J.: *Present Conditions of Mining in the District of Vladivostok, Siberia*, xii, 592–602.
Boron: effect in hardening cobalt-chromium alloys, 577.
Boston & Montana Reduction Department, Great Falls, Mont.: gas-fired furnace, 801.
Brinell method of measuring hardness, 356.
BRINSMADE, R. B.: *Discussion on Our National Resources and Our Federal Government*, 633–640.
Bronze: metallography, 847.
Brunt iron-mine, Mesabi Range: ore-concentration, 47.
Bullion: *sampling for assay*, 853–882.
By-Product Coke (ANDREWS), xiii, 851–852.
By-product coke-ovens:
comparison with other types, 157, 179.
cost of plant, 158.

Calamine-deposits: Siberia, 593.
CAMP, J. M.: *The Methods of the United States Steel Corporation for the Commercial Sampling and Analysis of Pig-Iron*, xiii, 220–249.
Campbell: heat-loss by radiation, 259.
CAMPBELL, WILLIAM; *Microstructure of Iron and Steel*, xii, 489–512.
Notes on the Metallography of Alloys, xiii, 825–850.
Canada: iron-ore deposits, [36].
Canadian Copper Co.: experiments with pulverized fuel, [793].
Cananea Consolidated Copper Co., Cananea, Mexico:
furnace data, 795.
oil-fired furnace, 796.

Digitized by Google

CANBY, R. C.: *Development of the American Water-Jacket Lead Blast-Furnace*, xiii, 736–740.
Carbon:
determining in pig-iron, 237.
effect in hardening cobalt-chromium alloys, 577.
effect in oxidation of steel in the furnace, 550.
effect in removal of phosphorus in the furnace, 561.
effect of high, on charcoal iron, 314–355.
Carbon compounds formed in slags, 565.
Carbon-iron equilibrium diagram:
Ruff's, 430–477.
Wittorff's, 478–488.
Carbonizing-materials: *action of*, 371–421.
Case-hardened steel:
microstructure, 511.
relations of hardness and depth of carbonization, 356–369.
Cast-iron; see Iron.
Centigrade temperatures: *conversion to Fahrenheit*, 883–888.
Centrifugal Machines for Ore-Grading and Ore-Concentrating (VIVIAN), xii, 676–682; *Discussion* (DEKALB), 682–683.
Charcoal-iron: see Iron.
Chauvenet, Regis: analysis of titaniferous iron-ore, [14].
Chemistry of the Reduction Processes in Use at Anaconda, Mont. (LAIST), xi, 806–817.
China:
iron-ore deposits: Kiang-su province, 34.
Hu-pei province, 27–37.
Chromium: *alloys with cobalt and other metals*, 573–579.
Chromium-deposits: Siberia, Vladivostok district, [600].
Cleveland-Cliffs Iron Co.:
forms of reports on drilling and sampling, 82.
surveying and sampling drill-holes, 69–90.
Clinton Iron-Ore Deposits in Kentucky and Tennessee (WHINERY), xii. 25–26; *Discussion* (MACFARLANE), 889.
Coal:
analysis (Diamondville, Wyo.), 784.
compressing, for coke-manufacture, [172].
derivation of bituminous matter, [154].
effect of oxygen on coking qualities, 155, 185.
Pocahontas: yield of coke, 159.
preparation for coking, 171.
yield of by-products, [155].
Coal-deposits: Siberia: Vladivostok district, 600.
Cobalt: *alloys with chromium and other metals*, 573–579.
Coke:
by-product, 181–183.
efficiency in the cupola, 201–210.
manufacture, 154–200.
production, U. S., 1907–1911, 156, 178, 182.
structure for foundry and furnace use, 155, 184, 191.
Coke-manufacture: 154–200.
at low temperatures, [193].
disposal of gas, 160.
gas production, 160, 168, 179.
growth of the industry, 181.
power required for various operations, 167.
recovery of by-products, 167, 174, 178.
value of by-products, 189.
yield of by-products, 178.

Digitized by Google

Coke-ovens:
application of heat in different types, 163.
comparison of types, 157, 179.
costs of construction and operation, 179, 180.
horizontal flue regenerative type, 164.
horizontal vs. vertical flues, 193.
storage of heat in walls, 194.
tonnage per oven, 158.
producer gas for heating, 183.
silica brick in construction, 160.
Colorado: titaniferous iron-ore deposit, Boulder county, 14-25.
Compass: Maas drill-hole, 72.
Compressed-air transmission: *solution of D'Arcy's formula*, 705.
Concentrating: see Ore-treatment.
Concentration of Iron-Ores, (HANSELL), ix, 37-57; *Discussion* (D'INVILLIERS), 60; (GRAMMER), 64-67; (JOHNSON), 59-60; (MOORE), 59; (PAYNE), 60-64; (RICHARDS), 67-68; (SPILSBURY), 58-59.
Constitution and Melting-Points of a Series of Copper-Slags (FULTON), xi, 751-780.
Converter reactions (copper), 814.
Copper:
addition to rail-steel, 279.
determining in pig-iron, 247.
effect of alloying with titanium, 293.
freezing- and melting-point curves, 766.
Copper-deposits:
Korea, 606, 611.
Siberia: Vladivostock district, 599.
Copper-furnace gases, Kyshtim, Russia: analysis, 788.
Copper-furnace reactions, 819.
Copper-furnaces:
development of reverberatory, 781-805.
Korean, 609.
Copper-mines:
Korea: Huchang, 606.
Kapsan, 611.
Copper-ores:
analyses, 820-823.
sulphatizing-roasting, 818-824.
Copper-slags: *Constitution and Melting-Points*, 751-780.
Copper-smelting:
analyses of materials used and produced, 788.
chemistry of reduction processes at Anaconda, 806-817.
data of furnaces, 786.
development of the reverberatory furnace, 781-805.
oil fuel, 790.
Copper-tin-antimony alloys: metallography, 834.
Costs:
coke-ovens: construction and operation, 179, 180.
iron-ore concentration, 57.
Crushing: ores, 43.
Cupola practice:
air per ton of iron, 214.
blast pressure, 213.
coke: effect of moisture, 191.
ratio to iron, 216.

Digitized by Google

Cupola practice.—Continued.
coke: sizing, 192.
structure for foundry use, 155, 184, 191.
heat calculations, 201–210.
height of bed, 215.
melting iron, 211–219.
oxygen: effect in melting iron, 213, 218.
temperature, 212, 214,
tuyere arrangement, 212.
Cupolas:
Balliot, [210].
fuel-efficiency, 201–210.
Curtis, J. S.: deflection of drill-holes, [79],
Cuyuna, Minn., iron-ore district:
correlation with Mesabi district, [4].
geology, 3.

D'Arcy's formula for transmission of compressed air: *graphic solution*, 705.
Day, Arthur L., and Allen, E. T.: melting-points of minerals, [135].
Deoxidation in the electric furnace, 561.
Dephosphorization in the electric furnace, 560.
Desulphurization in the electric furnace, 562.
Development of the American Water-Jacket Lead Blast-Furnace (CANBY), xiii, 736–740; *Discussion* (DRAKE), 890.
Development of the Parkes Process in the United States (EURICH), xiii, 741–750.
Development of the Reverberatory Furnace for Smelting Copper-Ores (MATHEWSON), xiii, 781–805.
DEWEY, FREDERICK P.: *The Sampling of Gold-Bullion*, xii, 853–879; *Discussion* (KELLER), 879–882.
Diamond-drill holes: *surveying and sampling*, 69–90.
D'INVILLIERS, E. V.: *Discussions: on The Concentration of Iron-Ores*, 60.
on The Manufacture of Coke, 185–186.
Divorcing annealing: *influence on the mechanical properties of low-carbon steel*, 513–547.
DRAKE, FRANCIS, *Discussion on Development of the American Water-Jacket Lead Blast-Furnace*, 890.
Drill-holes:
changing direction by wedging, 69.
surveying and sampling, 69–90.
Drown's method for determining silicon in pig-iron, 222.
DUMBLE, E. T.: *The Occurrence of Gold in the Eocene Deposits of Texas*, xiii, 588–591.

Effect of Alumina on Blast-Furnace Slags (JOHNSON, J. E., JR.), ix, 123–132; *Discussion* (FULTON), 133–135; (J. E. JOHNSON), 135; (W. MCA. JOHNSON), 139–142; (KEMP), 135–136; (PORTER), 139–142.
Effect of High Carbon on the Quality of Charcoal-Iron (JOHNSON, J. E., JR.), xi, 314–345; *Discussion* (HOWE), 346–350; (MOLDENKE), 350–354; (STOUGHTON), 354–355.
Efficiency-records: *utility in the manufacture of iron*, 143–153.
Eichhoff: solubility-curve of iron oxide, 559.
EILERS, A.: *Notes on Bag-Filtration Plants*, xiii, 708–735.
Electric furnace: *heat-losses*, 569–572.
Electric Heating and the Removal of Phosphorus from Iron (GREENE), xii, 548–556.
Engineering Experiment Station: experiments on transfer of heat through walls, 165.
EURICH, ERNST F.: *The Development of the Parkes Process in the United States*, xiii, 741–750.
Experience in the Use of Water-Power (MYRICK), xii, 701–704.

Digitized by Google

Fahrenheit temperatures: *conversion to centigrade*, 883–888.
Ferro-titanium: manufacture, 293.
Filtration: *bag-filtration plants*, 708–735.
Fires in Metalliferous Mines (YOUNG), xii, 644–662.
FIRMSTONE, F.: *Discussion on Blowing-In a Blast-Furnace*, 115–119.
FITZGERALD, F. A. J.: *Heat-Losses in Furnaces*, xiii, 569–572.
Florin, Jean: determining deviation in drill-holes, 71.
Flotation: *Wood process*, 686–701.
Flue-dust:
  bag-filtration, 708–735.
  (copper): analysis, Kyshtim, Russia, 789.
Ford: determining manganese in pig-iron, 235.
  determining silicon in pig-iron, 223.
Fuel Efficiency of the Cupola-Furnace (PORTER), xii, 201–210.
Fuel-gas, Great Falls smelter, Montana: analysis, 788.
FULTON, CHARLES H.: *The Constitution and Melting-Points of a Series of Copper-Slags*, xi, 751–780.
  Discussions: on The Effect of Alumina in Blast-Furnace Slags, 133–135.
    on The Manufacture of Coke, 196–197.
Fume recovery, 708–735.
Function of Slag in Electric Steel-Refining (AMBERG), xii, 557–568.
Furnace reactions (copper), 806–817.
Furnaces:
  copper: *development of the reverberatory*, 781–805.
  electric: *action in steel-refining*, 559.
    heat-losses, 569–572.
  electric arc, 486.
  lead: *development of the water-jacket*, 736–740.
  open-hearth: divided spout for tapping, 250–262.
    stream-controller, 251.
    Phœnix Iron Co., Phœnixville, Pa., 250–262.
  Wedge multiple-hearth; tests, 818–824.

Gabbro: analyses, 16.
Gas-fired copper-furnaces, 785–793.
Gases:
  Copper-furnace, Kyshtim, Russia: analysis, 788.
  formation in cupola, 206.
  fuel, Great Falls, Mont., copper-smelter: analysis, 788.
  illuminating: production in coke-plants, 160, 168, 179, 181.
  producer: from wood, Kyshtim, Russia: analysis, 788.
    use in heating coke-ovens, 183.
  transmission of heat from, 187.
  yield from by-product coke-ovens, 177–179.
Gee centrifugal ore-concentrating machine, 676.
Geology:
  China: Yangtze river, 32.
  Colorado: Boulder county iron-ore deposits, 14.
  Korea: northern, 604.
  Minnesota: Cuyuna iron-ore district, 3.
  ultimate source of metals, 663–675.
Gilbreth, Frank; motion study, [146].
Gold-bullion: *sampling*, 853–882.
Gold-deposits:
  Siberia: Vladivostok district, 693.
  Texas: *Eocene*, 588–591.

Digitized by Google

Gold-mines:
Korea; Oriental Consolidated Mining Co., [611].
Suan, [611],
Gold-silver deposits: Korea, 610, 611.
Goltra, E. G.: ore-concentration process, 47, 58.
GRAMMER, F. LOUIS: *Discussions: on The Concentration of Iron-Ores*, 64–67.
on The Manufacture of Coke, 198–200.
Graphic Solution of D'Arcy's Formula for the Transmission of Compressed Air in Pipes (HERZ), xiii, 705–707.
GREENE, ALBERT E.: *Electric Heating and the Removal of Phosphorus from Iron*, xii, 548–556.

HAAS, HERBERT: *Discussion on Temperature Conversion Tables*, 888.
HANSELL, N. V.: *The Concentration of Iron-Ores*, ix, 37–57.
Hardness: methods of measuring, 356, 368.
HAYNES, ELWOOD; *Alloys of Cobalt with Chromium and Other Metals*, ix, 573–579,
Discussion on Recent Developments in the Inspection of Steel Rails, 280.
Heat:
application in coke-ovens, 163.
radiation vs. convection in coking, 195.
transfer through walls, 165.
transmission from gases, 187.
Heat calculations, 201–210.
HERZ, NATHANIEL: *A Graphic Solution for D'Arcy's Formula for the Transmission of Compressed Air in Pipes*, xiii, 705–707.
HIBBARD, H. D.: *Discussions: on The Action of Various Commercial Carbonizing-Materials*, 421.
on Notes on Ruff's Carbon-Iron Equilibrium Diagram, 475.
on Recent Developments in the Inspection of Steel Rails, 278, 280.
Hillebrand, W. F.: analytical methods for rocks, [16].
Hoffman, John I.: diamond-drilling and bore-hole surveying, [69].
Hogampo copper-deposits, Korea, [606].
Hollister iron-mine, Crystal Falls, Mich.: dry ore-concentration, 47.
HOWARD, J. J.: *Discussion on Blowing-In a Blast-Furnace*, 113–115.
HOWE, HENRY M.: *Notes on Ruff's Carbon-Iron Equilibrium Diagram*, viii, 430–474.
Discussion on The Effect of High Carbon on the Quality of Charcoal-Iron, 346–350.
HOWE, HENRY M., and LEVY, ARTHUR G.; *Influence of Divorcing Annealing on the Mechanical Properties of Low-Carbon Steel*, xii, 513–547.
Huchang copper-mines, Korea, 606.
HUNT, ROBERT W.: *Recent Developments in the Inspection of Steel Rails*, viii, 269–281.
Hydrogen: effect on coking qualities of coal, 186.

Illuminating gas: production in coke-plants, 160, 168, 181.
Influence of Divorcing Annealing on the Mechanical Properties of Low-Carbon Steel (HOWE and LEVY), xii, 513–547.
International Smelting & Refining Co., Tooele, Utah: reverberatory copper-furnace, 805.
Iridium-deposits: Siberia: Vladivostok district, [600].
Iron:
analyses: charcoal, 339, 341, 494.
ingot, 494.
puddled, 494.
cast: cleansing effect of titanium, 282–312.
testing-machine for bars, 329.

Digitized by Google

Iron.—Continued.
charcoal: *effect of high carbon on quality*, 314-345.
grading-card for pig, 320.
classification, 490.
electric heating and removal of phosphorus, 548-556.
microstructure: busheled, 493.
charcoal-hearth, 492.
Dannemora, 492.
fagoted, 492.
puddled, 491.
wrought, 491.
oxidation to FeO: heat-units required, 202.
pig: analytical and sampling methods of U. S. Steel Corporation, 220-249.
titanium in, 291.
Iron-carbon equilibrium diagram:
Ruff's, 430-477.
Wittorff's, 478-488.
Iron manufacture: *utility of efficiency-records*, 143-153.
Iron-mines:
Adams, Cuyuna district, Minn., 11.
Barrows, Cuyuna district, Minn., 11.
Iron-ore:
analyses: Clinton, Kentucky, 26.
Cuyuna district, Minn., 8, 9.
titaniferous, 14, 23.
concentration, 37-68.
reduction in the electric furnace, 553.
removal of phosphorus, 548-556.
Iron-ore deposits:
Canada, [36].
China: Kiang-su province, 34.
Ta-yeh, Hu-pei province, 27-37.
Colorado: Boulder county (titaniferous), 14-25.
Kentucky and Tennessee: Clinton, 25-26.
Minnesota: Cuyuna district, 3-13.
New York: Adirondack region, 313.
Siberia: Vladivostok district, 599.
Utah: Iron Springs district, [36].
Iron-Ores of the South Range of the Cuyuna District, Minn. (ZAPFFE and BARROWS), viii, 3-13.
Iron oxide in slag: solubility curve, 559.

JENNINGS, E. P.: *Titaniferous Iron-Ore Deposit in Boulder County, Colo.*, xii, 14-25.
JOHNSON, J. E., JR.: *The Effect of Alumina in Blast-Furnace Slags*, ix, 123-132, 135.
The Effect of High Carbon on the Quality of Charcoal-Iron, xi, 314-345.
New Type of Blast-Furnace Construction, xiii, 91-104.
Discussions: on Alloys of Cobalt with Chromium and Other Metals, 579.
on Blowing-In a Blast-Furnace, 112-113.
on The Concentration of Iron-Ores, 59-60.
on Measurements and Relations of Hardness and Depth of Carbonization in Case-Hardened Steel, 368.
on Notes on Ruff's Carbon-Iron Equilibrium Diagram, 474, 477.
on Notes on Titanium and on the Cleansing Effect of Titanium on Cast-Iron, 312.
JOHNSON, W. MCA.: *Discussion on The Effect of Alumina in Blast-Furnace Slags*, 139-142.

Digitized by Google

Kapsan copper-mine, Korea, 611.
KELLER, EDWARD: *Discussion on The Sampling of Gold-Bullion*, 879–882.
KEMP, JAMES F.; *Discussions: On The Effect of Alumina in Blast-Furnace Slags*, 135–136.
on The Manufacture of Coke, 185, 198.
on Notes on Ruff's Carbon-Iron Equilibrium Diagram, 477.
on Notes on Titanium and on the Cleansing Effect of Titaninm on Cast-Iron, 313.
dike form of iron-ore deposits, Adirondack region, [24].
Kentucky:
Clinton iron-ore deposits, 25.
coal-deposits, 26.
oil-field, Wayne county, 25.
Korea:
copper-mine: Kapsan, [611].
gold-silver deposits, 610–611.
map: mineral deposits, 603.
trip through northern, 603–611.

LAIST, FREDERICK: *Chemistry of the Reduction Processes in Use at Anaconda, Mont.*, xi, 806–817.
Discussion on Recent Developments in the Inspection of Steel Rails, 279.
Lake Superior Iron & Chemical Co.: new type of blast-furnace construction, 97.
Lead:
desilvering: Balbach process, 741.
Parkes process, 741–750.
refining: elimination of zinc, 747.
smelting; water-jacket blast-furnace, 736–740.
recovery of fume, 708–735.
Lead-silver deposits: Siberia: Vladivostok district, 597.
Lead-tin-antimony alloys: metallography, 826.
Lead-zinc silver mines: Nertchinsk, Siberia, 598.
Leasing vs. sale of mineral lands, 612–643.
LEDOUX, ALBERT R.: *Discussion on Alloys of Cobal twith Chromium and Other Metals*, 579.
LEE, R. H.: *Discussion on Blowing-In a Blast-Furnace*, 119–121.
LEVY, ARTHUR G., and HOWE, HENRY M.: *Influence of Divorcing Annealing on the Mechanical Properties of Low-Carbon Steel*, xii, 513–547.
Lime: effect in blast-furnace slags, 125.
Liquids, miscibility of, 665.
LUCAS, F. E.: *The Manufacture of Coke*, ix, 170–180.

Maas drill-hole compass, 72.
MACCALLUM, N. E.: *Recent Developments in Open-Hearth Steel Practice*, xii, 250–262.
MACFARLANE, GRAHAM; *Discussion on Clinton Iron-Ore Deposits in Kentucky and Tennessee*, 889.
Magmas: differentiation theory of, 663.
Magnesia: effect in blast-furnace slags, 127.
Magnetic separation of iron-ores, 48.
Mallet: igneous fusion theory, [666].
Manganese: determining in pig-iron, 234.
Manganese-deposits: Siberia: Vladivostok district, [600].
Manufacture of Coke (BLAUVELT), ix, 154–169; (LUCAS), ix, 170–180: *Discussion* (BLAUVELT), 184, 185, 193–196: (D'INVILLIERS), 185–186: (FULTON), 196–197; (GRAMMER), 198–200; (KEMP), 185, 198; (MOLDENKE), 190–192; (PARKER), 189, 190; (SCHNIEWIND), 186–189, 194–195, 197: (STOEK), 193; (STOUGHTON), 192; (THROPP), 184.

Digitized by Google

Maps:
China: Ta-yeh, Hu-pei province, 30.
Korea (northern), 603.
Siberia (eastern): mineral deposits, 594.
Washington (State), 580.
MATHEWSON, E. P.: *The Development of the Reverberatory Furnace for Smelting Copper-Ores*, xiii, 781–805.
Measurements and Relations of Hardness and Depth of Carbonization in Case-Hardened Steel (AMMON), viii, 356–368; *Discussion* (JOHNSON), 368: (STOUGHTON), 369.
Melting Iron in the Cupola-Furnace (MOLDENKE), xii, 211–219.
Melting-points: apparatus for determining, 772.
Mesabi iron-ore district: correlation with Cuyuna district, Minn., [4].
Metallography of Alloys, 825–850.
Metallurgy; see names of metals.
Metals: anti-friction, bearing, babbitt, electrotype, etc. (see also Alloys):
composition, 827, 835, 845.
ultimate source, 663–675.
Methods of Preparing Basic Open-Hearth Steel for Castings (MILLER), xii, 263–268.
Methods of the United States Steel Corporation for the Commercial Sampling and Analysis of Pig-Iron (CAMP), xiii, 220–249.
Microstructure of Iron and Steel (CAMPBELL), xii, 489–512.
MILLER, H. F., JR.: *Methods of Preparing Basic Open-Hearth Steel for Castings*, xii, 263–268.
Mine-fires:
causes, 647.
fighting, 655.
legal regulations, Nevada, 657.
Metalliferous, 644–652.
prevention, 650.
proposed regulations, 658.
Mineral lands: leasing vs. sale, 612–643.
Minnesota: iron-ore deposits, Cuyuna district, 3–13.
Miscibility of liquids, 665.
Mold for sampling pig-iron, 222.
MOLDENKE, R.: *Melting Iron in the Cupola-Furnace*, xii, 211–219.
Discussions: on The Effect of High Carbon on the Quality of Charcoal Iron, 350–354.
on The Manufacture of Coke, 190–192.
Molybdenum: effect in cobalt-chromium alloys, 577.
MOORE, PHILIP N.: *Discussions: on Alloys of Cobalt with Chromium and Other Metals*, 577.
on the Concentration of Iron-Ores, 59.
Moose Mountain, Ltd.: concentrating-plant, Sellwood, Ont., 54.
MYRICK, C. M.: *An Experience in the Use of Water-Power*, xii, 701–704.

National Carbon Co., Cleveland, Ohio: visit to plant, xiv.
National Resources and Federal Government, Our, 612–643.
Nertchinsk, Siberia: lead-zinc-silver mines, 598.
New Type of Blast-Furnace Construction (JOHNSON), xiii, 91–104.
Nickel-steel:
case-hardening, 422–429.
constitutional diagram, 423.
Note on the Case-Hardening of Special Steels (SAUVEUR and REINHARDT), xi, 422–428: *Discussion* (ABBOTT), 428.
Notes on Bag-Filtration Plants (EILERS), xiii, 708–735.
Notes on the Metallography of Alloys (CAMPBELL), xiii, 825–850.

Digitized by Google

Notes on Ruff's Carbon-Iron Equilibrium Diagram (HOWE), viii, 430–474: *Discussion* (HIBBARD), 475: (JOHNSON), 474, 477: (KEMP), 477; (STOUGHTON), 475–476; (SWEETSER), 475–476.
Notes on Titanium and on the Cleansing Effect of Titanium on Cast-Iron (STOUGHTON), viii, 282–312: *Discussion* (JOHNSON), 312; (KEMP), 313; (STOUGHTON), 312.
Nozzle for pipe-line: reducing the size of, 703.

Occurrence of Gold in the Eocene Deposits of Texas (DUMBLE), xiii, 588–591.
Oils:
fuel for copper-smelting, 790.
petroleum deposits: Kentucky-Tennessee, 25.
Siberia: Vladivostok district, 602.
Oliver Iron Mining Co.: ore-concentration, 45, 63.
Open-hearth steel: *preparing for castings*, 263–268.
Ore-concentrating: see Ore-treatment.
Ore-deposits: see names of metals.
Ore-genesis: Ta-yeh iron-ore deposits, Hu-pei province, China, 34.
Ore-treatment:
concentrating: *centrifugal machines for*, 676–683.
Goltra process, 47, 58.
iron-ore, 37–68.
Wood flotation process, 686–701.
Woodbury system, 47.
crushing, 43.
jigging, 46.
magnetic separation, 48.
microscopic study, 40.
Ores: see names of metals.
Oriental Consolidated Mining Co.: gold-silver mines, Korea, [611].
Our National Resources and Our Federal Government (RAYMOND), xiii, 612-623; *Discussion* (BRINSMADE), 633–640; (RAYMOND), 641-643; (SMITH), 623–633; (WEPFER), 640–641.
Oxygen:
combination with coal, 197.
effect on calorific value of coal, 198.
effect on coking qualities of coal, 155, 185.
effect in the cupola, 213, 218.

PARKER, E. W.: *Discussion on The Manufacture of Coke*, 189-190.
Parkes process of desilvering lead: *development in the United States*, 741–750.
Parr, W. S.: coking at low temperatures, [193].
PAYNE, C. Q.: *Discussion on The Concentration of Iron-Ores*, 60–64.
Pennsylvania Steel Co.: concentrating-plant, Lebanon, Pa., 54.
Petroleum-deposits:
Kentucky-Tennessee, 25.
Siberia: Vladivostok district, 602.
Phœnix Iron Co., Phœnixville, Pa.: open-hearth steel-practice, 250–262.
Phosphor-bronze: metallography, 848.
Phosphorus:
determining in pig-iron, 229.
furnace reactions, 554.
limit in rail steel, 281.
oxidation in the steel furnace, 550.
removal from iron, 548–556.
Pig-iron: see Iron.
Pipe-line nozzle: reducing the size of, 703.

Digitized by Google

Pipe steel: analysis, 494.
Platinum-deposits: Siberia, [600].
PORTER, JOHN JERMAIN: *Fuel-Efficiency of the Cupola-Furnace*, xii, 201-210.
The Utility of Efficiency-Records in the Manufacture of Iron, xii, 143-153.
Discussion on The Effect of Alumina in Blast-Furnace Slags, 136-139.
Present Conditions of Mining in the District of Vladivostok, Siberia (BORDEAUX), xii, 592-602.
Proceedings of the 103d Meeting, Cleveland, Ohio, October, 1912, vii.
Producer-gas from wood, Kyshtim, Russia: analysis, 788.

Quicksilver-deposits: Siberia, Vladivostok district, [600].

Rail ingots: percentage of discard, 278.
Rails:
forms for reports of inspection, 273-277.
recent developments in inspection, 269-281.
RAYMOND, R. W.: *Our National Resources and our Federal Government*, xiii, 612-633, 641-643.
Read, T. T.: mineral resources of China, [27].
Recent Developments in the Inspection of Steel Rails (HUNT), viii, 269-277; *Discussion* (BLAUVELT), 278; (HAYNES), 280; (HIBBARD), 278; (HUNT), 278-281; (LAIST), 279; (SWEETSER), 281.
Recent Developments in Open-Hearth Steel-Practice (MACCALLUM), xii, 250-262.
REINHARDT, G. A., and SAUVEUR, ALBERT: *Note on the Case-Hardening of Special Steels*, xi, 422-428.
Retort coke-ovens; see Coke-ovens.
Reverberatory furnace reactions, 810.
Reverberatory furnaces (copper): *development at Anaconda, Mont.*, 781-805.
RICHARDS, R. H.: *Discussion on The Concentration of Iron-Ores*, 67-68.
Roasting: copper-ores and concentrates, 818-824.
Rock stresses, 666.
Rocks:
analyses: gabbro, Boulder county, Colo., 16.
quartz-monzonite, 807.
heat necessary for fusion, 671.

Saghalien Island:
coal-deposits, 601.
petroleum-deposits, 602.
St. Helen's Mining-District (WINCHELL), xii, 580-587.
Sak Ju gold-silver deposits, Korea, 610.
Salt: melting-point curve, 766.
Sampling:
diamond-drill holes, 69-90.
pig-iron: *methods of U. S. Steel Corporation*, 220-249.
Sampling of Gold-Bullion (DEWEY), xii, 853-879; *Discussion* (KELLER), 879-882.
SAUVEUR, ALBERT, and REINHARDT, G. A.: *Note on the Case-Hardening of Special Steels*, xi, 422-428.
SCHNIEWIND, F.: *Discussion on The Manufacture of Coke*, 186-189, 194, 195, 197.
producer-gas to heat coke-ovens, 183.
Scleroscope method of measuring hardness, 356, 368.
Seltzer, A. J.: Ta-yeh, China, iron-ore deposits, [28].
Separation, magnetic, of ores, 48.
Siberia:
map, mineral deposits, 594.
present conditions of mining, 592-602.

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Silica brick: in coke-oven construction, 160.
Silicon:
determining in pig-iron, 222.
effect in hardening cobalt-chromium alloys, 577.
effect in oxidation of phosphorus in the furnace, 549.
Silver:
separating from lead: Balbach process, 741.
Parkes process, 741–750.
Silver-gold deposits: Korea, 610–611.
Silver-lead deposits: Siberia; Vladivostok district, 597.
Silver-lead mines: Tetiuhe, Vladivostok district, Siberia, 597.
Slag:
analyses and composition, 136.
(copper) *constitution and melting-points*, 751–780.
effect of alumina in, 123–142.
effect of lime in, 125.
effect of magnesia in, 126.
function in electric steel-refining, 557–568.
influence of temperature of formation, 138.
ratios of lime, silica, and alumina, 128.
reactions in the electric furnace, 559.
Slag-tester: Johnson's, 139.
SMITH, GEORGE OTIS: *Discussion on Our National Resources and our Federal Government*, xi, 623–633.
SPILSBURY, E. G.: *Discussion on The Concentration of Iron-Ores*, 58, 59.
Steel:
case-hardened: *action of carbonizing-materials*, 371–421.
relations of hardness and depth of carbonization, 356–369.
case-hardening, 422–429, 511.
cementation, 511.
classification, 490.
copper additions, 279.
low-carbon: *influence of annealing on mechanical properties*, 513–547.
microstructure: alloy, 512.
axle, 504.
castings, 496.
high-carbon, 509.
low-carbon, 493.
medium-carbon, 496.
mild, 495.
pipe and skelp, 494.
rail, 505.
structural, 495.
tire, 508.
wrought, 494.
open-hearth: *preparing for castings*, 263–268.
pipe: analysis, 494.
rail: phosphorus limit, 281.
removal of phosphorus, 548–556.
special: *case-hardening*, 422–429.
titanium in, 292.
Steel-practice:
Bessemer: comparison with open-hearth, 551.
electric: deoxidation, 561.
dephosphorization, 560.

Digitized by Google

Steel-practice.—Continued.
electric: desulphurization, 562.
function of slag in refining, 557–568.
open-hearth: addition of alloys, 265.
influence of the slag, 264.
large vs. small heats, 256, 263.
loss of heat by radiation, 259.
pouring, 267.
recent developments, 250–262.
size of furnace, 263.
stream controller for divided spout, 251.
tap-hole and spout, 265.
tar as fuel, 192.
working, 266.
Steel rails: *Recent Developments in Inspection*, 269–281.
Stellite (chromium-cobalt-tungsten alloy), 574.
Steptoe Copper Co., McGill, Nev.:
furnace data, 786.
oil-fired furnace, 805.
record of operations, 790.
STEVENS, BLAMEY: *The Ultimate Source of Metals*, xiii, 663–675;
STOEK, H. H.: *Discussion on The Manufacture of Coke*, 193.
STOUGHTON, BRADLEY: *Notes on Titanium and on the Cleansing Effect of Titanium on Cast-Iron*, viii, 282–312.
Wittorff's Iron-Carbon Equilibrium Diagram, viii, 478–488.
Discussions: on The Effect of High Carbon on the Quality of Charcoal-Iron, 354–355.
on The Manufacture of Coke, 192.
on Measurements and Relations of Hardness and Depth of Carbonization in Case-Hardened Steel, 369.
on Notes on Ruff's Carbon-Iron Equilibrium Diagram, 475–476.
Suan gold-mine, Korea, 611.
Sulphatizing-Roasting of Copper-Ores and Concentrates (WEDGE), xiii, 818–824.
Sulphide ores: concentration by flotation, 686–701.
Sulphur: determining in pig-iron, 225.
Surveying and Sampling Diamond-Drill Holes (WHITE), xi, 69–90.
SWEETSER, R. H: *Blowing-In a Blast-Furnace*, ix, 105–112, 121–122.
Discussions: on The Effect of Alumina in Blast-Furnace Slags, 132–133.
on The Manufacture of Coke, 197.
on Notes on Ruff's Carbon-Iron Equilibrium Diagram, 475–476.
on Recent Developments in the Inspection of Steel Rails, 281.
Sydvaranger, Norway, iron-mines: magnetic separation of ore, 54.

Tar:
recovery from coke-oven gas, 174, 179.
use as fuel in open-hearth steel-manufacture, 192.
Ta-yeh Iron-Ore Deposits, Hu-pei Province, China (WELD), xii, 27–37.
Taylor, F. W.: shop management, [146].
Temperature Conversion Tables (WALDO), iii, 883–887: *Discussion* (HAAS), 888.
Tennessee:
Clinton iron-ore deposits, 25.
coal-deposits, 26.
Tetiuhe copper-silver-lead-zinc mine, Vladivostok district, Siberia, 597.
Texas: gold-deposits (Eocene), 588–591.
Thermometer scales: conversion tables, 883–887.
THROPP, JOSEPH E., JR.: *Discussion on The Manufacture of Coke*, 1841.

Digitized by Google

Thorium sulphate: solubility in water, 472.
Tin-antimony-copper alloys: metallography, 834.
Tin-lead-antimony alloys: metallography, 826.
Tin-zinc alloys: metallography, 846.
Titaniferous Iron-Ore Deposit in Boulder County, Colo. (JENNINGS), xii, 14-25.
Titanium:
bibliography, 306.
chemical properties, 289.
determining in pig-iron, 245.
history, 283.
in cast-iron, 295.
in non-ferrous metals, 293.
in pig-iron, 291.
in steel, 292.
manufacture, 293.
occurrence, 285.
physical properties, 287.
Tong Jom copper-deposits, Korea, [606].
Trip Through Northern Korea (TURNER), xiii, 603-611.
Tubes: collapsing strength, 72.
Tungsten: effect in alloys of cobalt and chromium, 574.
TURNER, HENRY W.: *A Trip Through Northern Korea*, xiii, 603-611.

Ultimate Source of Metals (STEVENS), xiii, 663-675.
United States Steel Corporation: methods of sampling and analyzing pig-iron, 220-249.
Utah: iron-ore deposits, Iron Springs district, [36].
Utility of Efficiency-Records in the Manufacture of Iron (PORTER), xii, 143-153.

VIVIAN, GODFREY C.: *Centrifugal Machines for Ore Grading and Ore-Concentrating*, xii, 676-682.
Vladivostok district, Siberia: *present conditions of mining*, 592-602.
Vogt, J. H. L.: dike form of iron-ore deposits, Norway, [24].
investigations of slags, [137].
origin of nickel-deposits, [665].

WALDO, LEONARD: *Temperature Conversion Tables*, iii, 883-887.
Walter's method of determining manganese in pig-iron, 234.
Washington (State):
St. Helen's mining district, 580-587.
sketch-map, 580.
Washoe smelter, Anaconda, Mont.:
chemistry of reduction processes, 806-817,
furnace data, 786.
Water-jacket blast-furnace: *development*, 736-740.
Water-power: *experience in the use of*, 701-704.
Wedge multiple-hearth furnace: tests, 818-824.
WEDGE, UTLEY: *The Sulphatizing-Roasting of Copper-Ores and Concentrates*, xiii, 818-824.
WELD, C. M.: *The Ta-yeh Iron-Ore Deposits, Hu-pei Province, China*, xii, 27-37.
WELLMAN, S. T.: *Discussion on Alloys of Cobalt with Chromium and Other Metals*, 578.
WEPFER, G. W.: *Discussion on Our National Resources and Our Federal Government*, 640-641.
Willis: geology of Yangtze gorges, China, [32].
WHINERY, S.: *Clinton Iron-Ore Deposits in Kentucky and Tennessee*, xii, 25-26.
White, David: effect of oxygen in coal, [198].

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WHITE, E. E.: *Surveying and Sampling Diamond-Drill Holes*, xi, 69–90.
WINCHELL, HORACE V.: *The St, Helens Mining-District*, xii, 580–587.
Wisconsin Steel Co.; ore-washing plant, Nashwauk, Minn., 46.
Wittorff's Iron-Carbon Equilibrium Diagram (STOUGHTON), viii, 478–488.
Woodbury system of ore-concentration, [47].
WOOD, HENRY E.: *The Wood Flotation Process*, xiii, 686–701.
Wrought-iron: see Iron.

YOUNG, GEORGE J.: *Fires in Metalliferous Mines*, xii, 644–662.

ZAPFFE, CARL: *Discussion on Alloys of Cobalt with Chromium and Other Metals*, 577.
ZAPFFE, CARL, and BARROWS, W. A., JR.: *Iron-Ores of the South Range of the Cuyuna District, Minnesota*, viii, 3–13.
Zinc-base alloys: metallography, 845.
Zinc-deposits: Siberia: Vladivostok district, 593.
Zinc-lead-copper-silver mine: Tetiuhe, Vladivostok district, Siberia, 597.

ERRATA.

PAGE.	LINE.	
641	5, 12.	For "1,000 sq. m." read "10,000 sq. m."
641	23.	After the words "60 cents" add, "from which the man buys his food."

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G22.8
A512
V.44
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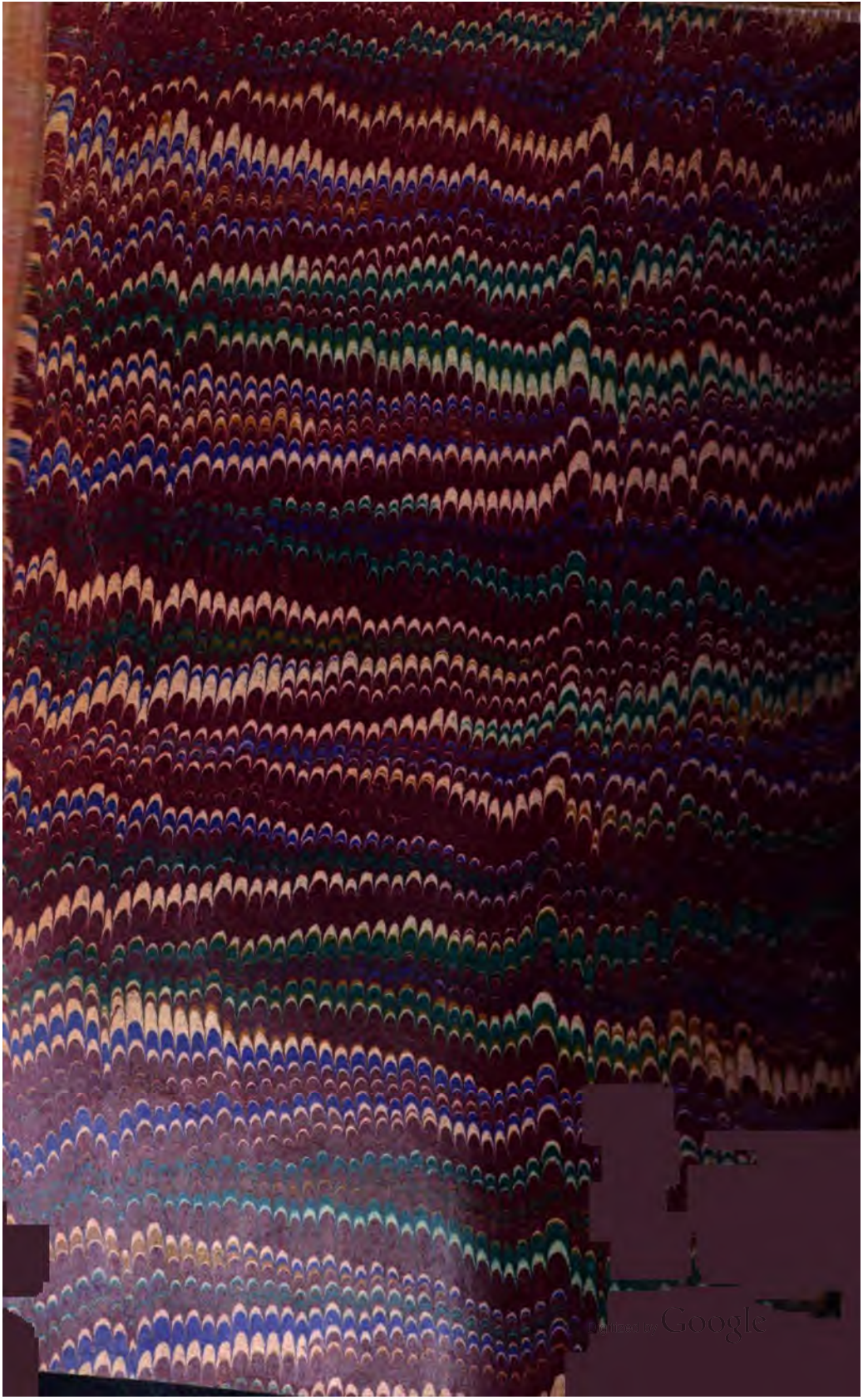
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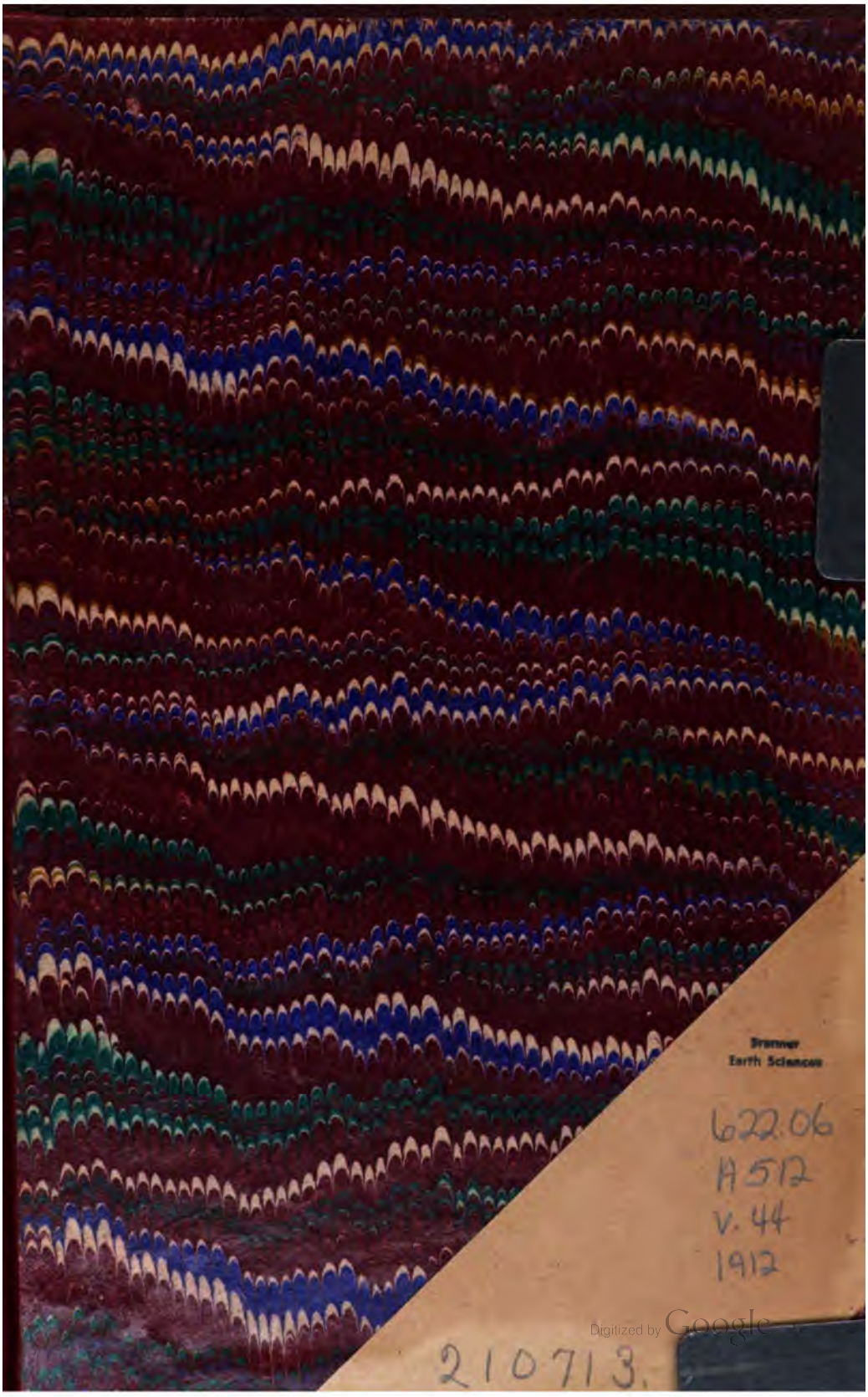
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